

A techno-economic evaluation of the production of hard coking coal from Tshikondeni coal discards

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Declaration

I, David Mark Powell, hereby declare that this thesis entitled:

A techno-economic evaluation of the production of hard coking coal from Tshikondeni Coal discards

which I submit to the North-West University in completion of the requirements for the degree of Philosophiae Doctor, is my own work except where acknowledged and has not been submitted to any other University before.

Signed at Potchefstroom on 8th of May 2015

Abstract

Tshikondeni Coal, operated by Exxaro Resources Limited on behalf of the owners, ArcelorMittal South Africa, is currently the only operational hard coking coal mine in South Africa. Therefore the yield from the plant needs to be maximised at the correct product specification, whilst losses to discard prevented as far as possible. The entire plant product is supplied to the ArcelorMittal South Africa's Van Der Bijl Park works. Any shortfall in production is made up for in costly imports from either Australia or New Zealand.

This thesis describes research work undertaken to:

- 1) Quantify the losses of product to discard of the process plant in terms of the dense medium cyclone and flotation circuits by efficiency testing, sizing analyses and discard washabilities. It was determined that the major contributors to the losses were in the -3mm size fraction for the dense medium cyclone and +1.0mm size fraction in the flotation circuit.
- 2) An extensive borehole drilling campaign was undertaken on the discard dumps on the mine to characterize the material on the dumps.
- 3) Samples were taken from the arising discard as produced by the normal operation of the plant and comprised of routine monthly discard samples, i.e. 1 tonne bulk samples from each of the operating shafts.
- 4) For the samples collected in 2) and 3) above, each were examined in detail by the application of appropriate technologies to effect upgrading by destoning, followed by milling and froth flotation. Solid-liquid separation and briquetting were also briefly examined.

It was found that the discard could be effectively destoned by two potential methods, firstly by dense medium separation alone, or secondly by removal of the +8mm material by screening followed by dense medium separation. Thereafter the material could be milled to -212 μ m prior to froth flotation to produce the final product.

A technical evaluation methodology was developed during the project which can be directly used on any given coal discards dump for future dump retreatment studies using the proposed stepwise approach.

The final yields obtained for the dump borehole samples after froth flotation were noted to be low at approximately 15% for a product specification of 14% ash content. The yields obtained on the monthly composite samples were found to be worse with yields of approximately 6% to 11% being obtained.

A plant design, based upon the application of the proposed technical evaluation methodology, was developed by using the main plant once ROM sources had been depleted and modified to include milling as well as the addition of a filter press and briquetting plant. A financial evaluation was performed in accordance with the Exxaro Project Evaluation Tool developed by the Corporate Finance Department. This revealed that a positive net present value (NPV) was achieved and that both the internal rate of return and the modified internal rate of return (which allows for risk within the project technically and also the project phase) were well above the hurdle rate set. Moreover, the payback period was found to be just under 2 years relative to a project life of 10 years. Simplification of the flowsheet to allow for only destoning using screening and dense medium separation also produced favourable results at similar yield of approximately 10% and the relatively minor modifications to the plant would require marginally over 1 year for payback. Yield in this instance may also be improved upon by optimization of the flotation circuit to process the fines fraction

Unfortunately the project was not pursued further due to lack of strategic fit for both Exxaro and ArcelorMittal South Africa. However, the dump could be considered for sale to a third party for implementation of the project. Given the lack of industrial development, and consequentially the high unemployment rate, in this remote and rural area of South Africa, the latter could make for a good local economic development project for the local community.

An alternative approach was also considered in parallel to the above work: could the plant be reconfigured to improve the yields? Five alternative plant circuits were examined and it was found that the overall yield could be improved by between 3%

and 4%. This section of the work was scheduled to proceed to a bankable feasibility study level including the detailed design, but was ultimately stopped due to the global economic down turn which began in 2008.

During 2015 the mine will enter the mine closure phase and the dump will be rehabilitated according to the mine closure plan, which requires covering in topsoil and grassing. This will maintain the integrity of the material on the dump and allow for future processing of the dump should the project be reconsidered.

Keywords used in the thesis:

Coal Preparation, coal discards, screening, dense medium separation, milling, froth flotation, dewatering, briquetting, economic evaluation

Dedication

This work is dedicated to my mother Joyce Powell, for the many sacrifices she made under extremely difficult conditions so that I could get a good education.

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To Him who deserves all honour and glory, for the strength to continue when things did not happen as they “should” in a project of this nature.

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Chapter 1: Introduction

As the chapter title suggests, an introduction to the project is given and begins with an overview of the location and geology of Exxaro's Tshikondeni Coal, the only operational hard coking coal mine in South Africa at the time of writing.

The plant and product qualities are briefly described before the motivation behind the project is introduced: being the only source of hard coking coal it was necessary to determine whether any losses of the plant product to the discard streams can be economically recovered.

The scope of the work is then described and concludes with the layout of the thesis, which was compiled in a non-traditional format as will be seen as the reader progresses through the various chapters.

1.1 Tshikondeni Coal

Tshikondeni Coal is a colliery operated by Exxaro Resources Limited located in the north eastern corner of the Limpopo Province close to the Kruger National Park and within the Eastern Zoutpansberg coalfield as shown in Figure 1.1, which also depicts the mine in relation to the major coalfields of South Africa, Figure 1.2 shows the mine more specifically in terms of the local environment.

Tshikondeni Coal is the only source of hard coking coal in South Africa and all the product is supplied to the local steel producer ArcelorMittal South Africa works located in Van Der Bijl Park. Any shortfall in production results in ArcelorMittal South Africa having to import the difference from either Australia or New Zealand.

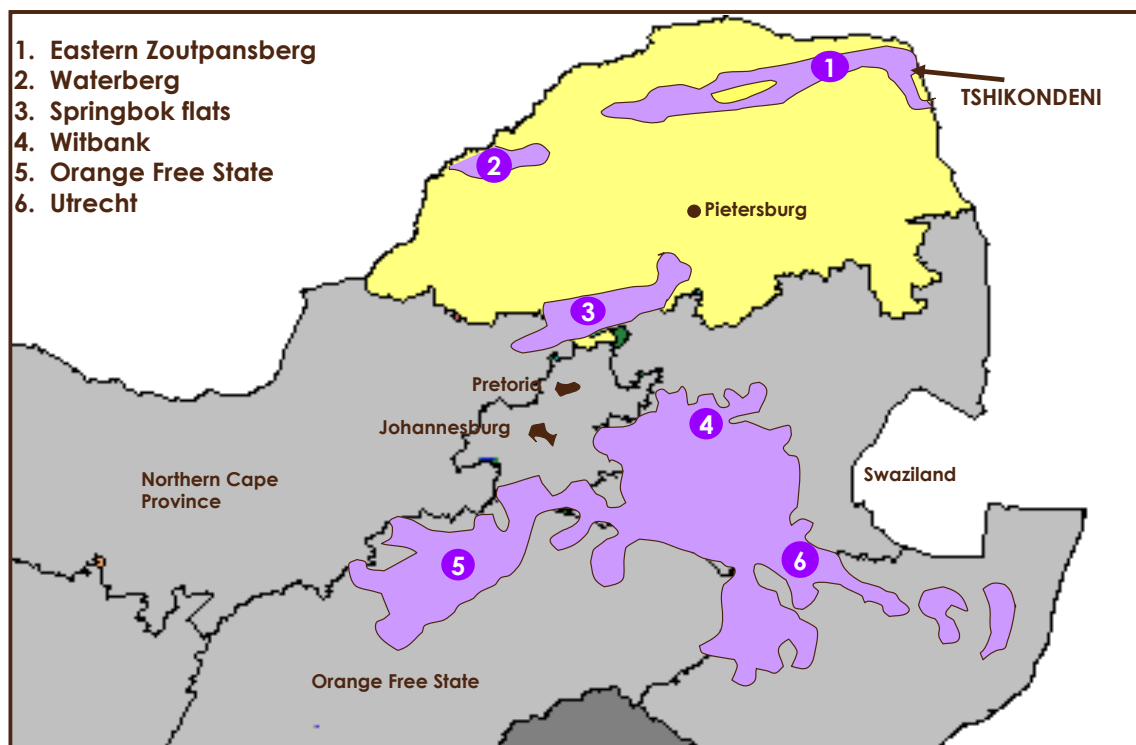


Figure 1.1: Tshikondeni general location in relation to the major Coalfields of South Africa (Gregory, 2007)

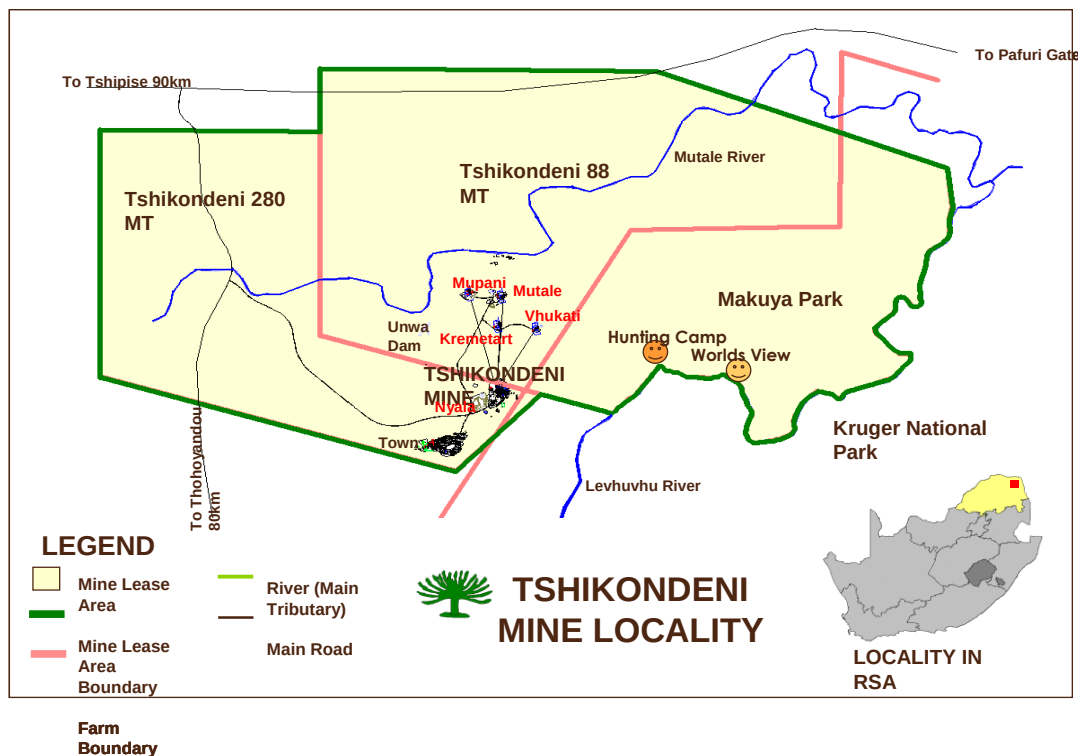


Figure 1.2: Tshikondeni immediate locality (Gregory, 2007)

In terms of geology (Gregory, 2007), the area around Tshikondeni was tectonically active during various geological periods. It overlies the Pretorezoic Limpopo Mobile Belt consisting of extensive east-north-east trending linear zones of high-grade metamorphic tectonites. The north-eastern margin of the Kaapvaal Kraton was down faulted into a graben-type structure into which the pre-Karoo Soutpansberg Group was deposited. This faulting continued during the deposition of the Karoo sediments and was again reactivated in the post Karoo times, resulting in a very complex structural setting. Two main fault systems have been identified, one trending east-north-east and the other north-north-west.

The Karoo series was faulted into a series of horst and graben blocks with displacements exceeding 1000m in some cases. The faults have a listric nature and are all normal faults, which is an indication of extensional tectonics as shown in Figure 1.3.

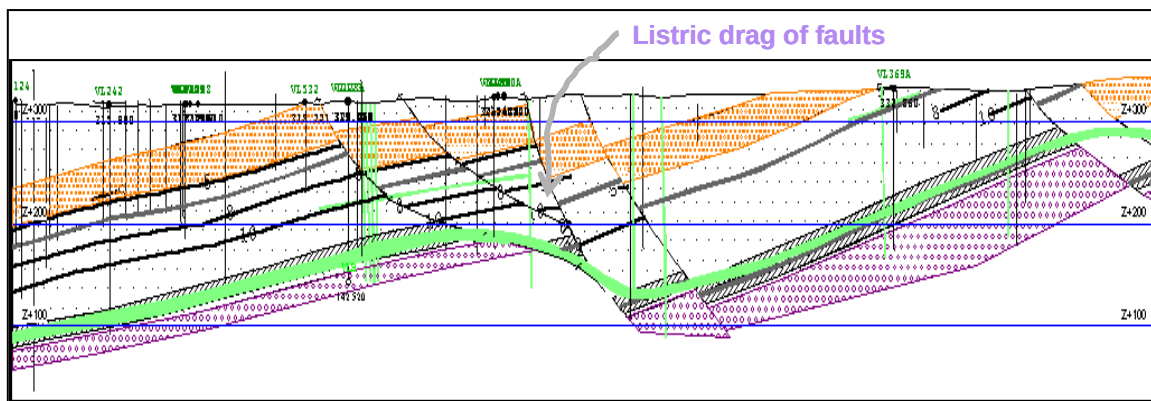


Figure 1.3: Listric faults at Tshikondeni (Gregory, 2007)

Gregory (2007) further indicated that the blocks between faults are tilted and the dip of the strata varies between 2° and 18° , but increases to 22° near the faults. Due to the dip of the coal seam, the mine operates from surface to a maximum depth of 400 meters below surface. The coal seams were deposited similarly to those of the Vryheid formation and are locally known as the Madzaringwe formation. It is a challenging coal mine as far as mining conditions are concerned due to the extensive faulting and intensity of dolerite dykes and sills in the area. The sills are up to 30m thick and the dykes are up to 20m wide. Contact aureoles vary, but can reach up to 60m on either side of the dykes, but average influence is about the thickness of the dyke. The rocks are fine to crystalline and green to dark green and gray. Yellowish chill, flow banding and flow laminations are common. Some dykes have spinifex textures. Brecciation and fracturing at the contacts are common, as are slickensides associated with small scale faulting of less than 2 metres.

The mine currently produces run of mine material from three underground shafts named Vhukati, Mutale and Goni, the product of which are transported by road trucks to the processing plant. The run of mine from each of the shafts is processed separately due to marked differences in washability characteristics. Goni is the highest yielding of the shafts, producing yields in excess of 60% at the product specification of 14% ash, whilst Mutale gives the lowest yields at 45-50%, Vhukati typically gives yields in the order of 55%.

The first stage of processing is crushing and screening to give a plant feed of -13mm. Thereafter, it is fed into the main plant consisting of two identical modules

each treating up to 120tph of raw feed. The -13+1.4mm material is processed in 710mm dense medium cyclones, whilst the -1.4+0mm material is separated by means of froth flotation using standard coal reagents i.e. paraffin as the collector and a frother. The plant flowsheet is shown simplistically in Figure 1.4. The froth flotation section treats approximately 30% of the plant feed, and produces a clean coal in the order of 12% ash at yields of 70-80%. This allows for the dense medium cyclone, handling 70% of the plant feed, to produce a higher ash product and maximize yield. The two sections are combined to produce a final ash product specification of 14%. The typical final product specifications are shown in Table 1.1.

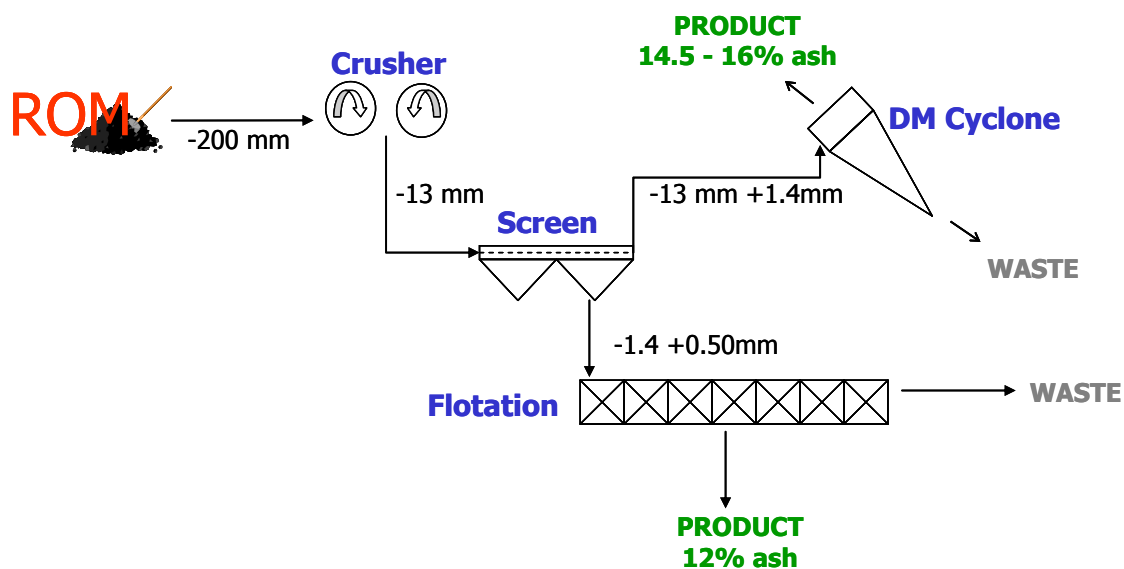


Figure 1.4: Tshikondeni plant simplified flowsheet

Table 1.1: Tshikondeni product specifications

Quality Parameter	Specification
Ash (% dry basis)	14.0
Volatile matter (% dry basis)	22.5-24.0
Sulphur (% dry basis)	<0.75
Phosphorus (% dry basis)	<0.048
Free swelling index	9
Roga index	80-90
Total Moisture (%)	<10%
Fines Content (% -0.5mm)	<30

Although the product is relatively high in ash compared to other sources of hard coking coal, the high free swelling and Roga indices are indicators of good coking properties. The typical petrographic and coking properties of the Tshikondeni

product are summarized in Table 1.2. The following section will give a brief overview of the main quality parameters applicable to a hard coking coal

Table 1.2: Typical petrographic and coking properties

Quality Parameter	Typical Value
<i>Total Sulphur (%) present as</i>	0.74
Pyritic Sulphur (%)	0.14
Sulphate Sulphur (%)	0.04
Organic Sulphur (%)	0.56
<i>Maceral Composition</i>	
Vitrinite (%)	80.9
Liptinite (%)	Not present
Reactive Semi-Fusinite (%)	2.6
Inertinite (%)	8.8
Mineral Matter (%)	7.7
<i>Petrographic Parameters</i>	
Reflectance of Vitrinite (max) (%)	1.42
Reflectance of Reactives (%)	1.42
Total Reactives (%)	81.54
Total Inerts (%)	18.46
Optimum Inerts (%)	12.62
Composition Balance Index	1.54
<i>Predicted Drum Indices</i>	
M ₁₀ Index	6.9
I ₁₀ Index	19.4
I ₂₀ Index	77.9
<i>Dilation Properties</i>	
Softening Temperature (°C)	389
Maximum Contraction Temperature (°C)	414
Maximum Dilation Temperature (°C)	491
Maximum Contraction (%)	26
Maximum Dilation (%)	162
Amplitude (%)	188
<i>Gieseler Fluidity</i>	
Initial Softening Temperature (°C)	420
Maximum Fluidity Temperature (°C)	469
Resolidification Temperature (°C)	505
Maximum Fluidity (ddpm)	2338

The clean coal product is stockpiled prior to dispatch by road trucks to the railway siding in Musina. Thereafter the coal is railed to ArcelorMittal South Africa's Van der Bijl Park works where it forms part of the blend used to produce the coke used in the blast furnaces. Other components of the blend are a semi soft coking coal product

sourced from Exxaro's Grooteegeluk mine and three imported hard coking coals, two from Australia and one from New Zealand (Ackerman, 2007)

1.1.1 Hard Coking Coal Quality Parameters

The use of coal in any application be it metallurgical, thermal or gasification, is driven by the overall qualities and properties of any given coal source. Based upon the quality information detailed in Table 1.1 and Table 1.2 above, it is appropriate at this point to briefly consider the important, and often specific, quality parameters related to hard coking coals.

1.1.1.1 Proximate Analysis

The proximate analysis comprises of the inherent moisture content, ash content, volatile matter content and fixed carbon. The latter is obtained by subtracting the first three from 100% and is considered as measure of the rank of the coal when converted to a dry mineral matter free basis (Horsfall, 1992). Bennie (1995) correctly noted that the proximate analysis does not give any indication as to the coking properties of a coal under consideration. However, it indicates the amount of contamination in the coal (ash content) and what will be lost during heating up in the coking process (inherent moisture and volatile matter). After the coking process, the proportion of ash in the coke will be higher than in the raw coal, and therefore should be as low as possible.

Osborne (1988) suggested that for every 1% increase in coke ash, an additional 15kg of coke is required to reduce 1 tonne of iron ore, the additional carbon being required to heat and melt the ash. Zimmerman (1979) further stated that an increase of 1% ash in the coke may reduce the blast furnace productivity by 2% to 3%. At an ash content of 14% the coal supplied to AMSA, the ash level of the Tshikondeni product is higher than other coking coals used by AMSA in their coking process. However, since it is blended with four other sources, the detrimental effects are to a large extent overcome.

Low volatile coals typically produce high strength cokes, though the associated expansion can cause damage to the oven. Medium volatile coals, like that produced

by Tshikondeni, generally produce cokes with good strength and may either expand or contract during the coking process. High volatile coals produce cokes of low strength and normally contract during the coking process, and are therefore not likely to damage the ovens (Zimmerman, 1979).

1.1.1.2 Ultimate Analysis, Phosphorus and Chlorine

The ultimate analysis is comprised of the total carbon (different to fixed carbon), hydrogen, inherent moisture, ash content, total sulphur and oxygen by difference. It is typically reported on a dry, ash free basis to avoid the inclusion of hydrogen and oxygen from these sources. Though not strictly part of the ultimate analysis, in terms of coke production, Zimmerman (1979) commented that it may be desirable to consider the phosphorus and chlorine contents.

The most important component of the coal is obviously carbon which is required for reduction. High oxygen content coals produce relatively weak cokes, and there is a strong correlation between coke quality and oxygen content (Zimmerman, 1979).

Roux (1995) reported that both sulphur and phosphorus are detrimental to the iron making process.

Sulphur exists in coal in three forms, namely pyritic (FeS_2), sulphates (such as gypsum CaSO_4) and organic compounds in which sulphur is combined with carbon in the coal. From Table 1.2 it is evident that the majority of the sulphur in the Tshikondeni product is of organic origin and would therefore be virtually impossible to remove by traditional coal preparation techniques. In the carbonization process 65-70% of the sulphur remains in the coke and 20% passes into the gas phase as hydrogen sulphide. This leads to the formation of segregated ferrous or manganese sulphides in the steel's structure which cause brittleness. For each 0.1% increase in sulphur in coke, an additional 3.4kg of coke per tonne of iron ore is required (Osborne, 1998).

Phosphorus, even in low proportions in coke, is detrimental due to it forming Fe_3P which is readily dissolved in iron, but in the presence of carbon, has a reduced

solubility level and creates brittleness in cast iron and weak zones in steels which form as bands during rolling (Osborne, 1988).

Zimmerman (1979) indicated that chlorine, which enters the coal as chlorides, may have a detrimental effect on the refractory linings of the coking ovens. Fortunately, most coals contain low chlorine contents at around 0.1%, though any new source of coal for a coke oven should be tested prior to use.

1.1.1.3 Free Swelling and Roga Indices

The Free Swelling Index is a rapid indicator of the coking properties of a coal. A value of 4 or higher generally indicates a coal with good coking properties, whilst below 4 would be considered weakly coking (Osborne 1988; Horsfall 1992; Zimmerman 1979).

The Roga Index indicates the caking (binding) ability with non-coking coals. The resulting index should be as high as possible, though for a coal to be deemed a coking coal, the index must exceed 80. For blending with other coking coals, a minimum value of 50 is required (Horsfall 1992).

The Tshikondeni product would be considered a coking coal based on the typical specification values presented in Table 1.1 above.

1.1.1.4 Plastic Properties

The fluidity of a coal under heating is measured by means of the Gieseler Plastometer. Fluidity is generally developed when the coal melts after heating at around 300°C with re-solidification taking place at about 500°C. The plastometer also measures the temperature range in which the coal is fluid. The fluidity is measured in dial divisions per minutes (ddpm) and a fluidity of 60-1000ddpm is considered suitable for coking. (Horsfall (1992), Osborne (1988)). The Tshikondeni product in Table 1.2 was noted to exhibit typical plastometer properties in terms of the softening and re-solidification temperatures (420°C and 505°C respectively), though the maximum fluidity was high at 2338ddpm. Plasticity is an important measure since based on the range of fluidity and temperature readings obtained, it indicates which coals may or may not be blended together (Zimmerman, 1979).

Changes in volume during the fluid state are measured in a dilatometer which shows the contraction/expansion behaviour of a given coal. The test can also indicate the temperatures required for the softening and maximum temperatures Osborne (1988), which from the results presented in Table 1.2 were noted to be similar to those reported for the Gieseler Plastometer test.

1.1.1.5 Petrographic Constituents

These petrographic constituents are determined by examining a highly polished coal specimen under the microscope using reflective light. The examination reveals the relative proportions of the macerals vitrinite, liptinite (typically not found in the Tshikondeni product), reactive semifusinite, inertinite as well as the mineral matter. A well-formed coke should ideally contain 75% reactives and 25% inerts, excessive deviation from this proportion in either direction results in a weak coke being formed. In terms of the results in Table 1.2, the reactives were higher than desirable at approximately 81.5% and the inerts correspondingly low at 18.5%.

The mean maximum reflective of the vitrinite in the sample is an indicator of rank, the higher the number the higher the rank. In non-coking coals, vitrinite would have a reflective value of less than 0.5%, whereas coking coals reside in the range between 0.5-2.0% and anthracites above 2.0%. The value for Tshikondeni is in the coking coal range at 1.42%. Falcon (2008) indicated that rank as determined by the reflectance of vitrinite can be used as a predictor of the possible use of coal in any given application as detailed in Figure 1.5 below, which clearly shows that the plant product from Tshikondeni Coal is in the prime coking coal category.

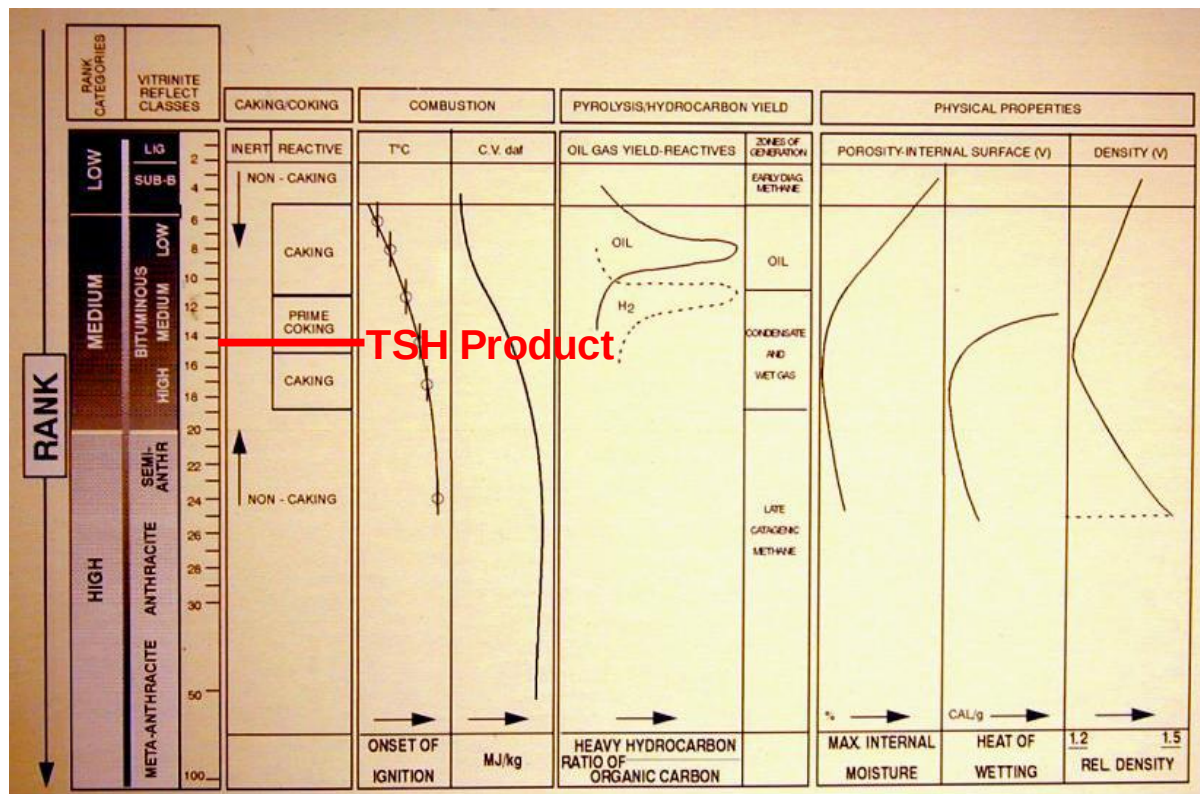


Figure 1.5: The prediction of coal usage based upon rank (Falcon, 2008)

Moreover, Osborne (1988) states that from the reflectance values and the vitrinite classes that the composition balance index and the strength index can be calculated which can be used to predict coke strength and that high levels of correlation between actual and predicted values can be achieved with certain coals. From Table 1.2 above, the predicted M_{10} Index of 6.9% suggests that the Tshikondeni Coal product should produce a strong coke.

1.2 Project Motivation, Project Objectives and Scope of Work

1.2.1 Project Motivation and Key Questions

Being the only operating hard coking coal mine in South Africa at the time of writing, the material produced by the mine could arguably be considered of strategic importance to South Africa since other sources of hard coking coal are imported. Exxaro Resources operates the mine on behalf of the owners, ArcelorMittal South Africa, at cost plus a 3% management fee based on the annual budget. It is therefore important that Exxaro manages the mine in a responsible way so as to extract as much value as possible for ArcelorMittal South Africa.

This project was therefore undertaken to determine whether or not the plant was at the time of writing being operated efficiently. Moreover, by examining the material in the discards dumps it would also be possible to see if the plant had been historically well managed since the planned testwork would show any deficiencies in historical operational performance.

Given this background, this body of work was undertaken to answer the following key questions:

- Is the existing coal preparation plant operating efficiently?
 - Is the dense medium cyclone circuit operating efficiently?
 - Is the froth flotation circuit operating efficiently?
- Though losses to discard are inherent in any coal preparation plant, can these losses be identified and quantified with respect to the plant at Tshikondeni Coal?
 - What, if any, value can be extracted from the original (old) discard dump?
 - What, if any, value can be extracted from the current operational discard dump?

- What, if any value, can be extracted from the fine coal material remaining in the old, disused, return process water dam (the “Visgat” dam)?
- What would be the most appropriate technology currently available be to recover the losses?
- Would a modification of the current plant be appropriate as a proactive measure to prevent losses from occurring?
- If a hard coking coal product cannot be produced from the plant discard, is there an alternative use for the discard?
 - Eskom (the South African state owned power utility)?
 - Gasification? From Figure 0.5 above, this option would appear to be ruled out.
 - Brickmaking? Small quantities of the discard have historically been sold to two local brick works at a nominal cost.

Depending on what, if any, product can be produced from the discard, can it be produced and supplied economically to both Exxaro and the end user given the remoteness of the Tshikondeni Coal location geographically?

1.2.2 Project Objectives

Based upon the discussion above in the project motivation and with respect to the key questions identified the following five objectives were set for the project:

1. To determine the operational efficiency of the plant and identify any possible losses of saleable product to the discard streams.
2. Characterization of the discards dumps and the naturally arising discards with the aim to recover any potentially saleable product lost by the plant to the dumps.
3. Perform an evaluation of various coal processing technologies for optimal flowsheet development for the design of a discards retreatment plant for the recovery of these losses.

4. Perform an economic evaluation of the proposed flowsheet for the discards retreatment plant.
5. To consider whether the existing plant could have been reconfigured to prevent or minimise the losses to the discards streams.

1.2.3 Scope of Work

The work undertaken for this project comprised of several phases, starting by examining the discard both that which had been deposited on the two discard dumps, and the arising discard.

The first section of the work was the determination of the losses to the discard and was extracted from data that was either readily available from the routine metallurgical accounting practices employed by Tshikondeni Coal, or from slight modifications to the routine test procedures.

Thereafter the starting point of the main body of the project was the characterization of the material on the two dumps and in the “Visgat” dam by drilling “boreholes” using an auger in a grid pattern. Thereafter additional work was performed on:

- Monthly discard composite samples.
- Samples of discard generated by the plant from the three operational shafts.
- A 20 tonne bulk sample taken from the old dump.

With the aim to produce a saleable product of 14% ash, a variety of techniques were examined, though the basic hypothesis was that the discard would need to be destoned followed by milling to liberate the values being lost prior to recovery by froth flotation. Product handling, by means of briquetting is also touched upon.

With respect to reconfiguring the plant to improve the yield, run of mine bulk samples from the two operational shafts were taken and used to simulate several potential plant reconfiguration scenarios that would enable an economic gain in the overall yield from the plant.

The samples were considered in terms of technology options for processing and as a result a methodology for the technical evaluation of a discard dump was proposed

based upon the work performed in a sequential manner. This methodology can be applied to any dump in any country and can be revised/updated as technology evolves to include any new developments within the field of coal preparation.

1.3 Thesis Layout

The layout of this thesis has taken a non-traditional format in that it is broken down into various sections starting with the quantification of the actual losses of product to the discard streams, followed by the characterization of the material on discard dumps, and then by the application of the different technologies explored with respect to recovery of a saleable product from the discard sources as per the proposed methodology mentioned above. Several of the thesis chapters could be considered as stand-alone reports and, where appropriate, therefore contain a literature survey related to the subject matter covered, the experimental details, results obtained and discussion thereof. Given that a non-traditional format was followed for this work, the overall flow of the thesis is illustrated in Figure 1.6 below, which excludes the appendices.

The scope of work undertaken entailed extensive testwork and therefore the detailed results relevant to any given chapter have been included as appendices to the thesis at the end of the thesis.

Following on from the conclusions and recommendations, the thesis closes with a retrospective on the candidate's contribution to engineering research.

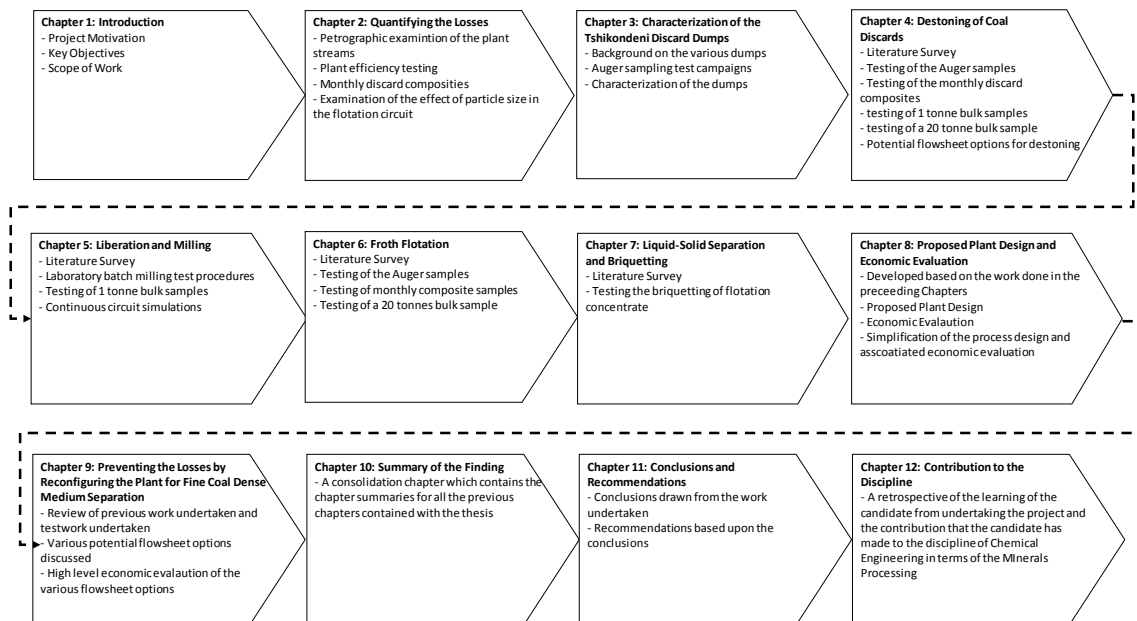


Figure 1.6: Flow diagram depicting the thesis layout

1.4 Chapter References

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Chapter 2: Quantifying the Losses

The first step of the work performed was to determine and quantify the losses of saleable coal to the discard streams.

An initial examination of the petrographic properties of the feed, product and discard was undertaken and the proportions of the macerals and the vitrinite classes for each plant stream determined. The discard was noted to contain approximately 47% vitrinite, which if recoverable would be suitable for supply as a hard coking coal product.

Plant efficiency tests by size revealed that the dense medium cyclone lost efficiency below 2.8mm and that any losses to the discard would tend to be to the finer fractions. The discard was also noted to contain a significant proportion of larger, slab like material.

The monthly discard composite samples were examined in terms of the washability of the +1.4mm material and it was noted that very little losses were being experienced at the lower density fractions. The flotation testing on the -1.4mm material showed losses of 7.2% in mass terms at an ash of 14%. However, subjecting the -1.4mm material to a float and sinks procedure revealed that approximately 40% of the mass could be recovered at an ash of 14% and relatively low separation relative density of the order of 1.52-1.53.

The flotation circuit efficiency was therefore further examined in terms of size, and several samples from each of the three shafts examined. It was found that the losses in this section were to the +0.6mm fraction.

2.1 Petrographic Examination of Tshikondeni Coal Plant Streams

As a precursor to the motivation for the project, a quality assessment of the feed to plant, product and discard streams was performed by Powell (2007) and summarised in Table 2.1.

Table 2.1: Feed to Plant, Product and Discard Qualities August 2007 (Powell, 2007)

Analysis	Feed to Plant	Plant Product	Plant Discard
Ash Content (% dry basis)	26.6	13.1	48.9
Volatile Matter (% dry basis)	19.8	22.5	15.9
Fixed Carbon (% dry basis)	53.3	64.4	35.1
Total Sulphur (% dry basis)	0.88	0.79	0.86
Vitrinite (%)	68.3	81.2	47.4
Liptinite (%)	-	-	-
Reactive Semifusinite (%)	5.5	2.5	11.8
Inertinite (%)	11.6	8.8	14.2
Minerals (%)	14.6	7.3	26.6
% V11 (1.10 to 1.19)	2	-	7
% V12 (1.20 to 1.29)	17	23	48
% V13 (1.30 to 1.39)	48	35	23
% V14 (1.40 to 1.49)	21	18	16
% V15 (1.50 to 1.59)	8	15	3
% V16 (1.60 to 1.69)	3	8	3
% V17 (1.70 to 1.79)	1	1	1
% RoV (max)	1.38	1.40	1.32

The proportion of vitrinite in the discard at 47.4% is higher than many of the coals produced from the Witbank, Highveld and Orange Free State coal fields as reported by Horsfall (1992) and therefore further consideration of reprocessing of the discard streams from the Tshikondeni plant would be justified. The reflectance of vitrinite (RoV) in the discard was determined to be 1.32 and is slightly less than that of the feed and product which were found to be 1.38 and 1.40 respectively.

The densimetric distribution of the vitrinite within the August 2007 plant discards streams was also determined during this exercise and the results are shown in Table 2.2. Further consideration of reprocessing of the discard streams from the Tshikondeni plant should be considered since the vitrinite contents of above 60%

would be expected after destoning at a relative density of 1.80. Vitrinite contents above 40% were noted up to a relative density of 1.60.

Table 2.2: Vitrinite content of the August 2007 plant discards by density fraction (Powell, 2007)

Float & Sinks Fraction	Vitrinite Content Fractional (%)	Vitrinite Content Cumulative (%)
Floats 1.30	91.6	91.6
1.30 - 1.35	83.3	89.0
1.35 - 1.40	82.8	88.1
1.40 - 1.45	70.9	85.7
1.45 - 1.50	64.7	82.6
1.50 - 1.55	60.4	79.2
1.55 - 1.60	43.4	76.2
1.60 - 1.65	38.5	72.2
1.65 - 1.70	29.4	66.7
1.70 - 1.75	24.6	63.1
1.75 - 1.80	21.6	61.8
Sinks 1.80	7.2	(47.4)

2.2 Plant Efficiency Testing

2.2.1 Routine Monthly Plant Efficiency Testing

The two DMS cyclone modules at the plant have been tested twice per month for several years and a substantial amount of data has been collected over the years. The tests follow the standard methodology applied to the majority of operating coal plants in South Africa and involves the collection of feed, product and discard samples which are subjected to float and sinks testing over the relative density range 1.30 to 1.80 in 0.05 intervals. and the ash content of each fraction determined. Thereafter the performance of the cyclone is calculated to give standard results such as the partition curve, ecart probable moyen (epm or Ep), yield, organic efficiency and misplaced material.

The effect of particle size on the efficiency of the cyclone was never considered and a test was conducted on 2nd August 2007 on the “A” module DMS cyclone. The objective of this test was to determine where the efficiency of the cyclone broke

away. All samples were subjected to a detailed sizing exercise and screened at +16mm, +11.2mm, +5.6mm, +2.8mm, +2.0mm, +1.4mm, +0.85mm and 0.6mm. It was considered impractical (and too lengthy) to subject each of these size fractions to individual washability testing and therefore size fractions were recombined such that size fractions of +11.2mm, -11.2+5.6mm, -5.6+2.8mm, -2.8mm+1.4mm and -1.4+0.6mm were subjected to separate efficiency tests and the performance of each size fraction was determined.

2.2.2 Plant Efficiency Test Results

The detailed results of the test described above are presented in Appendix 1. The results from the screening exercise are summarized in Table 2.3 and illustrated graphically in Figure 2.1. The DMS cyclone, being 710mm in diameter was operated at a pressure of 9D, equating 102kPa as per standard practice on most South African plants. The medium to coal to was approximately 3:1 as per plant design and the cyclone was fed at 70 tonnes per hour as per standard operating conditions. The medium differential, determined from the relative density of the underflow medium minus the overflow could not be determined due to plant layout and associated safety risks taking this reading. The plant uses a medium grade magnetite which was determined to be approximately 86% magnetic material and the -45µm fraction was approximately 88%, both these figures are in line with typical Tshikondeni historical results and therefore no cause for concern.

The feed material was noted to be coarser than expected at approximately 22% being retained on a 16mm test sieve, relative to the nominal -13mm that the plant is expected to process. The reason for this was found to be due to the crusher ahead of the plant being out of adjustment. A quirk of the plant design is only one stage of crushing using an Osborne Granulator (a type of rolling ring crusher), which requires extensive monitoring and regularly feeds the plant with oversize material. Examination of this fraction of the feed and discard clearly shows that the cyclone discard contained a significant proportion of +16mm material (~41%) which was also noted to be slab like as shown in section 4.3.3 in Figure 4.31.

The cyclone feed contained 3% of material finer than 1.4mm indicating good performance of the desliming screen. The -1.4mm material preferentially reported to the product stream as shown in Table 2.3, with the discard sample only containing 1% of -1.4mm particles.

Table 2.3: Summary of screening results

Screen	Feed			Product			Discard		
Size (mm)	Mass (%)	Cumulative Retained (%)	Percentage Passing Size	Mass (%)	Cumulative Retained (%)	Percentage Passing Size	Mass (%)	Cumulative Retained (%)	Percentage Passing Size
16	22.1	22.1	77.9	8.4	8.4	91.6	40.8	40.8	59.2
11.2	23.0	45.1	54.9	20.2	28.6	71.4	21.1	61.8	38.2
5.6	28.0	73.0	27.0	34.9	63.5	36.5	21.2	83.0	17.0
2.8	15.4	88.4	11.6	21.1	84.6	15.4	10.7	93.7	6.3
2	5.1	93.5	6.5	6.9	91.5	8.5	3.3	97.1	2.9
1.4	3.4	97.0	3.0	5.0	96.4	3.6	1.9	99.0	1.0
0.85	1.5	98.5	1.5	1.9	98.3	1.7	0.5	99.5	0.5
0.6	0.8	99.2	0.8	0.9	99.2	0.8	0.2	99.7	0.3
-0.6	0.8	-	-	0.8	-	-	0.3	-	-
Total	100.0	-	-	100.0	-	-	100.0	-	-

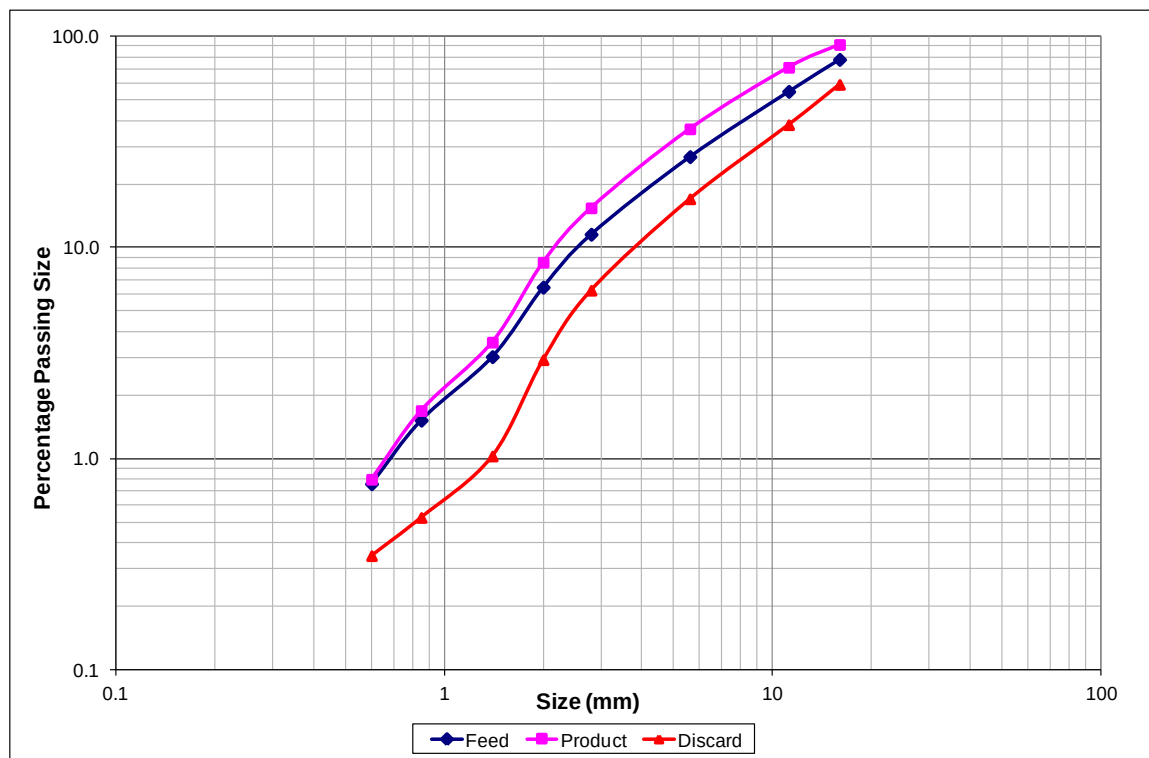


Figure 2.1: Screening results

Table 2.4 summarizes the results from the efficiency tests, whilst Figure 2.2 depicts a visual representation of the epm relative to particle size. The +11.2mm test produced the highest ash values of all the tests, with the discard being particularly high at 64%. Product ashes were noted to be higher than the typical product

specification of 14%, though this is normal. The flotation section of the plant typically produces a product of the order of 10-12% ash, and even at high reagent dosages does not reach 14% ash. Therefore the cyclone section can be run at a higher ash and the combined final product is achieved by manipulation of the operating density of the cyclone.

All the tests, except the -1.4+0.6mm material returned low epn values and as can be seen from Figure 2.2 the epn was noted to steeply “break away” below 2.8mm. Low organic efficiencies and high proportions of misplaced material, particularly floats (product) in sinks (discard) were observed for the -2.8+1.4mm and -1.4+0.6mm tests. Given that these two sizes accounted for 11.6 % of cyclone feed (approximately 8% of plant feed based on the typical 70% plant feed reporting to the cyclone section) a significant proportion of saleable product was being lost to discard. Moreover, the -1.4+0.6mm was also noted to have a high proportion of discard in the product and therefore obviously did not respond well in the cyclone as indicated by the high epn value of 0.045, though this size fraction only accounted for 2.3% of cyclone feed. The near density material in this fraction was also noted to be the highest of all the samples and would be an obvious contributor to the poor performance in terms of organic efficiency and misplacement of material. The overall plant near density material was the lowest, which was attributed to rounding errors when reconstituting the various fractions, it was typical of other tests though.

It is also noted that the cutpoint (d_{50}) of these two samples was different to the +11.2mm, -11.2+5.6mm and -5.6+2.8mm, which were all similar at 1.59 to 1.60.

Table 2.4: Summary of the efficiency tests

	Size Fraction (mm)					Overall Plant
	+11.2	-11.2+5.6	-5.6+2.8	-2.8+1.4	-1.4+0.6	
Feed Ash (%)	41.6	29.1	21.5	23.2	20.5	34.1
Product Ash (%)	20.6	19.4	17.3	14.9	13.8	18.5
Discard Ash (%)	64.2	54.7	56.1	52.7	47.3	64.0
Product Yield (%)	51.8	72.5	89.3	78.0	80.0	65.7
Feed Medium to Coal Ratio						3.0
Cyclone Feed Pressure (kPa)	-	-	-	-	-	102
Plant Operating Density (g/cm ³)	1.47	1.47	1.47	1.47	1.47	1.47
Calculated D50 (g/cm ³)	1.59	1.59	1.60	1.63	1.54	1.61
Density Offset (g/cm ³)	0.12	0.12	0.13	0.16	0.07	0.14
Epn	0.003	0.005	0.002	0.020	0.045	0.011
Organic Efficiency	98.9	95.3	99.0	93.5	93.2	98.6
Near Density Material (%)	18.1	17.7	17.1	13.7	24.1	14.8
Misplaced Material:						
Sink in Float (%)	0.9	0.4	0.2	0.6	7.7	0.2
Float in Sink (%)	0.7	2.3	1.7	5.6	5.1	2.2
Total Misplaced (%)	1.6	2.7	1.8	6.2	12.8	2.4

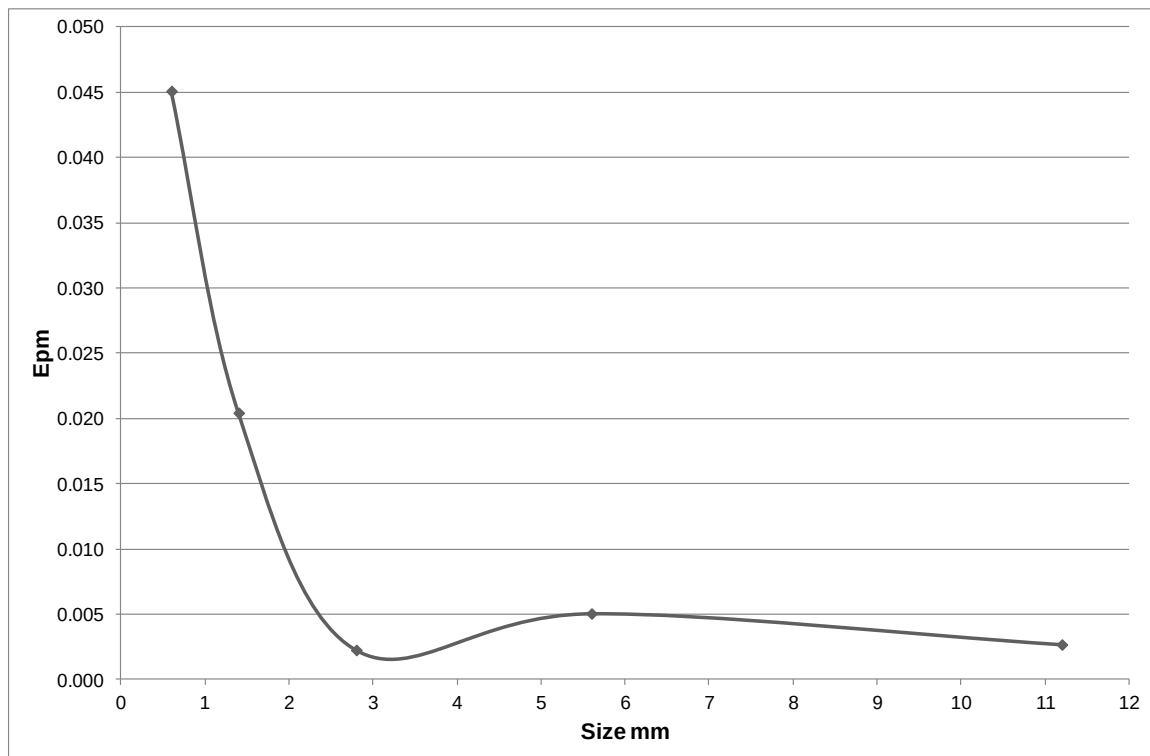


Figure 2.2: Epm relative to particle size

The “break away” in terms of Epm below 2.8mm is in line with what would be expected from a Multotec cyclone treating this size material at the feed rate, operating medium density and pressure (Bekker, 2008).

2.3 Monthly Discard Composite

2.3.1 Monthly Discard Composite Test Procedure

In addition to routine plant efficiency testing, a monthly (and feed) composite sample is generated by the plant laboratory. The sample is made up from representative splits of the hourly samples to form a daily sample, which is composited for a week. The weekly samples are then composited for the monthly sample and a standard washability test performed on the +1.4mm material over the relative density range 1.30 to 1.80 in 0.05 increments, with the quality of the individual density fractions being determined for yield and ash.

The -1.4mm material was initially subjected to a bulk rougher flotation test lasting 4 minutes, which was subsequently changed to a flotation rate test in late 2007. The

bulk results indicated that the discard material did not contain any saleable (14% ash material), though since the test looks at the overall floatability, some on grade material may be released in the early stages of the test and hence the decision to change from a bulk test to a rate test. These tests were conducted by using 500g of the discard sample in a 1.8 litre Denver cell using tap water, the slurry which had a pH of 6.5 was agitated for 2 minutes prior to the addition of 0.26mL of the premixed reagent comprising of 85% paraffin/15% Betafroth BTN3 frother used to replicate the reagent suite used in the flotation circuit and conditioned for 2 minutes prior to floating. Concentrates were collected at timed intervals of 0-30 seconds, 30-60 seconds, 60-120 seconds and 120-240 seconds. The four concentrates and the tailings were then weighed and the ash content determined.

From 2009 and in the light of the work described in the next section, the -1.4mm discard was also subjected to a washability test to account for what may be recoverable in a dense medium process since a fair proportion of the material was determined to be larger than 0.6mm and theoretically recoverable by this process route. This test is somewhat difficult to perform accurately since the mesh used in the basket used in the float and sinks operation was 0.5mm, and therefore losses of finer material would have been incurred during the test. However, it would give a fair representation of the losses of the +0.6mm material relative to the flotation test.

2.3.2 Monthly Discard Composite Results

The washability data from a typical monthly discard composite for the +1.4mm size fraction are presented in Table 2.5. From this data it is obvious there is very little saleable product being lost into this sample with only 0.3% of the material being at an ash content of 13.0% at a relative density of 1.30, whereas only 1.1% of the material is recoverable at a relative density of 1.35 and an out of specification ash level of 15.5%. These results would therefore appear to concur with those determined from the plant efficiency by size exercise.

Table 2.5: +1.4mm Discard washability

Relative Density	Yield (%)		Ash (%)	
	Fractional	Cumulative	Fractional	Cumulative
1.30	0.3	0.3	13.0	13.0
1.35	0.8	1.1	16.5	15.5
1.40	1.6	2.8	17.6	16.7
1.45	1.6	4.4	22.6	18.9
1.50	1.7	6.1	25.2	20.7
1.55	2.2	8.2	26.2	22.1
1.60	2.7	10.9	29.5	23.9
1.65	3.8	14.7	36.0	27.1
1.70	1.0	15.7	37.9	27.7
1.75	2.2	17.9	39.7	29.2
1.80	3.2	21.1	44.3	31.5
SINK 1.80	78.9	-	71.6	-
TOTAL :	100.0	100.0	63.1	63.1

The results of the -1.4mm material flotation rate test are presented in Table 2.6:

Table 2.6: -1.4mm Discard flotation rate test

Flotation Fraction	Yield (%)		Ash (%)	
	Fractional	Cumulative	Fractional	Cumulative
Concentrate 1 (30s)	2.8	2.8	13.7	13.7
Concentrate 2 (60s)	4.4	7.2	14.3	14.1
Concentrate 3 (120s)	1.3	8.5	23.9	15.6
Concentrate 4 (240s)	0.8	9.3	35.0	17.2
Tailings	90.7	-	46.2	-
Total	100.0	100.0	43.5	43.5

Here it is shown that 7.2% of the discard could be recovered at an ash value of 14.1%, which was essentially on grade and tied in closely with the work efficiency test on the -1.4+0.6mm material which indicated misplacement of product to the discard (5% mass at 13.8% ash). However, the washability results revealed a very different picture as indicated in Table 2.7. By using a gravity separation technique, approximately 40% of the discard would be recoverable at 14% ash at a relative density in the order of 1.52 to 1.53. This increase is most likely attributable to the fundamental difference in how the two techniques separate and that the coarser material in the discard would respond better to a gravity process rather than a surface properties process, i.e. the gravitational force (or centrifugal force when

moving around a bubble) exceeds the binding energy between the particles and bubbles to be lifted out of the slurry by the air bubbles. This was also confirmed by the work performed in the following section.

Table 2.7: -1.4+0.6mm Discard washability results

Relative Density	Yield (%)		Ash (%)	
	Fractional	Cumulative	Fractional	Cumulative
1.30	4.3	4.3	7.4	7.4
1.35	10.2	14.5	8.3	8.0
1.40	10.8	25.2	12.2	9.8
1.45	6.3	31.6	18.8	11.6
1.50	6.4	37.9	21.5	13.3
1.55	4.9	42.8	26.0	14.7
1.60	6.6	49.4	34.2	17.3
1.65	1.1	50.6	44.2	17.9
1.70	1.4	52.0	58.6	19.0
1.75	1.1	53.1	59.3	19.9
1.80	0.9	54.0	60.7	20.5
SINK 1.80	46.0	-	68.4	-
TOTAL :	100.0	100.0	71.6	42.6

2.4 Examination of the Effect of Particle Size in the Flotation Circuit

2.4.1 Examination of Size Test Procedure

An examination of the effect of size on the flotation circuit performance was undertaken over several weeks from the end of July 2008 to the end of August 2008 with the view to determine the losses of coarse material to flotation tailings. A total of 10 sets of samples comprising of the flotation feed, product and discard were taken from each of the three operational shafts at the mine viz. Mutale (3 individual samples), Vhukati (3 individual samples) and Goni (4 individual samples).

The respective samples were screened at 1.4mm, 1.0mm, 0.5mm, 0.3mm and 0.212mm. In addition to the mass distributions per size class, thereafter each size class was analyzed for ash content and the ash balance calculation used to determine yield to flotation product per size class.

2.4.2 Examination of Size Test Results

2.4.2.1 Mutale Shaft

The detailed results are presented in Appendix 1. The data from the three Mutale samples are summarized in Table 2.8 and Table 2.9 for the overall yield (by ash balance) and the individual sizes respectively. The feed, product and tailings from the three tests were each noted to be similar in quality with yields of approximately 57% to 63% being achieved.

Table 2.8: Overall yield and ash results (Mutale)

Date	Feed Ash (%)	Product Ash (%)	Tails Ash (%)	Yield (%)
23/07/2008	24.8	11.8	42.3	57.3
15/08/2008	24.6	11.1	44.1	59.1
21/08/2008	22.9	11.1	42.5	62.5

Table 2.9: Yield and ash results per size fraction (Mutale)

Date	Size	Fractional Sizing (%)			Cumulative Passing Sizing (%)			Ash (%)			Yield (%)
		Feed	Product	Tailings	Feed	Product	Tailings	Feed	Product	Tailings	
23/07/2008	+1.4mm	12.8	0.7	56.1	100.0	100.0	100.0	36.3	9.0	37.5	4.2
	-1.4+1.0mm	13.3	4.9	8.8	87.2	99.3	43.9	32.5	6.3	33.0	1.9
	-1.0+0.5mm	20.5	30.9	17.5	73.9	94.3	35.1	24.6	7.0	40.5	47.5
	-0.5+0.3mm	14.3	31.7	6.3	53.4	63.4	17.6	21.8	12.1	56.1	78.0
	-0.3+0.212mm	7.5	22.5	2.2	39.2	31.7	11.2	24.3	16.6	68.2	85.1
	-0.212mm	31.7	9.2	9.0	31.7	9.2	9.0	18.6	18.3	68.7	99.4
15/08/2008	+1.4mm	19.5	4.3	31.4	100.0	100.0	100.0	26.4	7.9	29.8	15.6
	-1.4+1.0mm	20.3	10.7	25.9	80.5	95.7	68.6	24.0	6.9	37.5	44.0
	-1.0+0.5mm	29.4	25.8	26.5	60.2	85.0	42.8	23.2	8.7	47.8	62.7
	-0.5+0.3mm	18.1	23.7	9.0	30.8	59.2	16.3	20.3	11.0	70.4	84.2
	-0.3+0.212mm	7.6	16.1	3.5	12.7	35.5	7.3	24.9	12.3	80.8	81.6
	-0.212mm	5.1	19.5	3.9	5.1	19.5	3.9	42.7	16.7	84.1	61.5
20/08/2008	+1.4mm	15.4	2.8	42.1	100.0	100.0	100.0	29.2	7.0	34.1	18.2
	-1.4+1.0mm	16.9	8.1	24.0	84.6	97.2	57.9	24.4	6.3	37.0	41.0
	-1.0+0.5mm	30.7	23.6	22.8	67.7	89.1	33.9	20.1	8.0	49.3	70.6
	-0.5+0.3mm	14.5	23.2	6.4	37.1	65.5	11.1	18.1	10.9	67.3	87.1
	-0.3+0.212mm	8.2	15.3	2.5	22.6	42.3	4.7	21.5	12.9	79.3	87.1
	-0.212mm	14.4	27.0	2.2	14.4	27.0	2.2	25.9	14.9	79.0	82.9

From Table 2.9 it is evident that the coarser material does not float well as seen in the lower yields experienced, particularly in the +1.4mm and +1.0mm size fractions due to particle gravitational forces exceeds bubble binding energy, and also the high proportion of material retained on these two screens in the tailings fraction. Yields in the finer fractions were higher, as would be expected along with higher ashes. The ability to use a collector to enhance hydrophobicity on the coarse particles would also have been negatively affected by the presence of the surface active unfine particles.

2.4.2.2 Vhukati Shaft

The Vhukati results are summarized in Table 2.10 and Table 2.11. The overall yields were noted to be better than Mutale due to the overall better quality of material from this section of the mine, though the yields were variable,

Table 2.10: Overall yield and ash results (Vhukati)

Date	Feed Ash (%)	Product Ash (%)	Tails Ash (%)	Yield (%)
28/07/2008	19.2	10.0	38.9	68.3
12/08/2008	23.5	12.0	31.9	42.1
21/08/2008	23.8	10.6	39.6	54.6

Table 2.11: Yield and ash results per size fraction (Vhukati)

Date	Size	Fractional Sizing (%)			Cumulative Passing Sizing (%)			Ash (%)			Yield (%)
		Feed	Product	Tailings	Feed	Product	Tailings	Feed	Product	Tailings	
28/07/2008	+1.4mm	15.6	1.7	71.6	100.0	100.0	100.0	27.6	6.8	34.3	24.4
	-1.4+1.0mm	23.7	9.9	6.9	84.4	98.3	28.4	23.7	7.0	30.2	28.0
	-1.0+0.5mm	26.7	32.5	10.6	60.7	88.4	21.5	14.6	8.3	45.0	82.8
	-0.5+0.3mm	17.2	26.0	3.8	33.9	55.9	10.9	14.8	11.3	61.7	93.1
	-0.3+0.212mm	4.1	9.9	2.4	16.7	30.0	7.1	14.1	13.4	68.6	98.7
	-0.212mm	12.6	20.0	4.6	12.6	20.0	4.6	17.5	11.3	73.9	90.1
12/08/2008	+1.4mm	20.2	3.3	53.7	100.0	100.0	100.0	25.7	8.4	26.9	6.3
	-1.4+1.0mm	20.2	4.9	19.8	79.8	96.7	46.3	21.7	7.6	31.2	40.2
	-1.0+0.5mm	33.8	29.9	20.6	59.6	91.8	26.4	22.8	8.9	43.0	59.1
	-0.5+0.3mm	15.0	22.8	3.8	25.9	61.9	5.8	22.0	11.0	26.9	31.2
	-0.3+0.212mm	6.0	17.2	0.9	10.9	39.0	2.0	21.3	13.3	66.3	85.0
	-0.212mm	4.9	21.8	1.1	4.9	21.8	1.1	34.3	17.7	74.6	70.9
21/08/2008	+1.4mm	21.1	4.1	36.7	100.0	100.0	100.0	26.0	6.8	29.8	16.6
	-1.4+1.0mm	17.0	9.0	23.6	78.9	95.9	63.3	21.7	6.7	31.3	39.1
	-1.0+0.5mm	30.7	25.5	25.2	61.9	86.9	39.6	21.1	8.3	45.3	65.4
	-0.5+0.3mm	14.9	25.8	8.0	31.3	61.4	14.5	20.4	10.2	62.3	80.5
	-0.3+0.212mm	5.8	16.6	3.0	16.4	35.6	6.5	24.5	11.9	71.5	78.9
	-0.212mm	10.6	19.0	3.5	10.6	19.0	3.5	34.6	15.8	77.4	69.4

Table 2.11 also showed that there were high losses of material to the tailings in the coarser sizes, particularly the sample of the 28th July, where some 71% of the material was retained on the +1.4mm screen. This however could be due to a poor sample since the stream is not very easy to sample in practice.

2.4.2.3 Goni Shaft

The four tests from Goni are summarized in Table 2.12 and Table 2.13. Goni being the best of the three reserves at the mine returned the highest yields. Again it was observed that losses of material to the tailing occurred in the coarser sizes and very high yields were achieved from the finer fractions.

Table 2.12: Overall yield and ash results (Goni)

Date	Feed Ash (%)	Product Ash (%)	Tails Ash (%)	Yield (%)
24/07/2008	17.8	10.0	46.2	78.5
06/08/2008	22.4	10.0	43.3	62.7
08/08/2008	17.8	12.4	44.5	83.2
15/08/2008	22.7	9.4	40.5	57.3

Table 2.13: Yield and ash results per size fraction (Goni)

Date	Size	Fractional Sizing (%)			Cumulative Passing Sizing (%)			Ash (%)			Yield (%)
		Feed	Product	Tailings	Feed	Product	Tailings	Feed	Product	Tailings	
24/07/2008	+1.4mm	9.5	0.9	37.7	100.0	100.0	100.0	19.7	7.2	35.6	56.0
	-1.4+1.0mm	15.0	7.8	23.3	90.5	99.1	62.3	18.0	5.8	41.9	66.2
	-1.0+0.5mm	36.5	33.8	23.4	75.5	91.3	39.0	15.4	7.2	50.6	81.1
	-0.5+0.3mm	14.4	35.6	8.1	39.0	57.5	15.6	11.6	10.0	69.3	97.3
	-0.3+0.212mm	13.1	11.0	3.5	24.6	21.8	7.5	24.3	10.6	74.8	78.7
	-0.212mm	11.5	10.8	4.0	11.5	10.8	4.0	23.9	21.6	74.1	95.6
06/08/2008	+1.4mm	12.0	2.9	50.7	100.0	100.0	100.0	26.4	7.0	35.6	32.3
	-1.4+1.0mm	23.0	8.8	20.9	88.0	97.1	49.3	22.5	7.3	41.9	55.9
	-1.0+0.5mm	34.3	36.4	18.2	65.0	88.3	28.4	19.7	8.4	50.6	73.0
	-0.5+0.3mm	16.7	20.3	6.2	30.7	51.9	10.2	18.1	9.6	69.3	85.8
	-0.3+0.212mm	7.7	10.6	1.6	14.0	31.6	3.9	23.7	10.4	74.8	79.3
	-0.212mm	6.3	21.0	2.3	6.3	21.0	2.3	38.6	14.4	74.1	59.5
08/08/2008	+1.4mm	6.4	2.4	46.3	100.0	100.0	100.0	22.3	8.8	38.3	54.2
	-1.4+1.0mm	8.7	8.8	15.1	93.6	97.6	53.7	15.7	7.0	33.2	67.0
	-1.0+0.5mm	26.1	24.5	21.0	85.0	88.8	38.7	14.6	9.8	47.4	87.2
	-0.5+0.3mm	22.5	23.1	8.4	58.8	64.2	17.7	16.3	12.0	62.7	91.5
	-0.3+0.212mm	15.5	15.4	4.4	36.3	41.1	9.3	14.1	12.9	66.5	97.8
	-0.212mm	20.9	25.7	4.8	20.9	25.7	4.8	25.9	17.3	74.1	84.9
15 08 2008	+1.4mm	13.9	1.6	24.6	100.0	100.0	100.0	27.1	9.8	32.1	22.3
	-1.4+1.0mm	18.3	7.4	33.2	86.1	98.4	75.4	24.7	6.9	29.5	21.0
	-1.0+0.5mm	27.0	29.8	24.7	67.9	91.1	42.2	20.7	8.0	44.1	64.8
	-0.5+0.3mm	16.8	23.4	8.1	40.8	61.3	17.5	18.6	8.7	62.8	81.6
	-0.3+0.212mm	9.0	11.6	3.9	24.1	37.8	9.4	18.6	9.6	72.5	85.6
	-0.212mm	15.0	26.3	5.5	15.0	26.3	5.5	26.8	12.4	73.6	76.5

From this section of the work it was evident that the plant design is inadequate and that significant losses of material to the flotation tailings were being experienced. If this and the poor dense medium response of the material below 2.8mm is considered, it is evident that losses to discard could be minimized by:

1. Reconfiguration of the plant as discussed in depth later on in Chapter 9. This would therefore essentially split this study into two sections, the first part examining the recovery of material in the dump with the section looking at ways of preventing losses to the dump.
2. A more detailed consideration given to coarse coal flotation.

With respect to point 2 above, Nicol (1984) indicated that coarse coal flotation is possible though specific conditions needing to be met as reported by Klassen and Mokrousov (1963)

- a. by an increase in the hydrophobicity of the surface by an appropriate collector addition,

- b. a controlled reduction of the pulp density (note that Tshikondeni floats at approximately 10% solids compared to other South African operations where 5-7% solids is the norm),
- c. the particles need to be lifted by appropriately sized bubbles,
- d. a reduction in the agitation intensity,
- e. an increase in the aeration rate by precipitation,
- f. providing additional hydrostatic lift by the creation of a rising pulp medium,
- g. removal of excess slimes (which would require plant modification)
- h. formation of a quiescent, and
- i. careful removal of the froth

Bensley and Nicol (1983) examined the effect of the mechanical variables on the flotation of coal and concluded that the poor performance of coarse particle flotation was not the inability of the coal to float, but rather the inability to float the coarse middlings material. This implied that surface chemistry of the particle is intermediate between that of a hydrophobic coal and a hydrophilic mineral. To overcome this, a non-conventional collector would be required. This conclusion was borne out by Yoon et al (2002), albeit on a Korean anthracite, whereby high product yields were obtained at particle sizes of up to 3mm using a specially formulated collector.

Therefore the practicality of coarse coal flotation needs to be further explored, though extensive work would be required not only from alteration of the flotation circuit to meet the requirements listed, but also to determine whether the specialist collectors can be obtained cost effectively.

2.5 Chapter Summary

The initial petrographic examination of the plant discard revealed that it contained approximately 47% vitrinite with a reflectance value of 1.32, which if economically recoverable would be suitable for supply as a hard coking coal product. At relative densities of up to 1.60 the fractional vitrinite content was found to exceed 40%.

Plant efficiency tests by size revealed that the dense medium cyclone lost efficiency below 2.8mm and that any losses to the discard would tend to be to the finer fractions. The discard was also noted to contain a significant proportion of larger, slab like material. The cyclone feed contained 3% of material finer than 1.4mm indicating good performance of the desliming screen.

The monthly discard composite samples were examined in terms of the washability of the +1.4mm material and it was noted that very little losses were being experienced at the lower density fractions. The flotation testing on the -1.4mm material showed losses of 7.2% in mass terms at an ash of 14%. However, subjecting the -1.4mm material to a float and sinks procedure revealed that approximately 40% of the mass could be recovered at an ash of 14% and relatively low separation relative density in the order of 1.52-1.53.

The flotation circuit efficiency was therefore further examined in terms of size, and several samples from each of the three shafts examined. It was found that the losses in this section were to the +0.6mm fraction. This was attributed to the gravitational forces of the particle (centrifugal when passing a bubble) being greater than the binding energy of the bubble. The ability to use a collector to enhance hydrophobicity on the coarse particles would also have been negatively affected by the presence of the surface active ultrafine particles.

From this chapter it is evident that losses occurred at the finer sizes of the dense medium section and coarser sizes of the flotation section of the plant. Discard retreatment or reconfiguration of the plant circuit could therefore improve the overall recovery of hard coking coal at Tshikondeni.

2.6 Chapter References

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Chapter 3: Characterization of the Tshikondeni Discard Dumps

In this chapter the qualities of the discard dumps on the mine are determined and the three separate dumps, the old dump, the current dump, which was operational at the time of writing, and an old return water dam were subjected to a detail characterization exercise in which a total of 30 auger borehole cores were drilled, with each separate core being made up of several 4.5m lengths to allow for special information to be generated with respect to the in-situ dump qualities.

A total of approximately 4.3 million tonnes of coal was determined to be present between the two dumps and one dam, with in-situ ash values of between 50-55% for the coarse (+0.5mm) material and 30-35% for the finer (-0.5mm) material.

A detailed characterization methodology is explained and the results for the in-situ characterization presented. Further work to upgrade the material found in the dumps and dam was performed using both dense medium separation and flotation after milling, the procedures employed and results obtained from these techniques are discussed in detail in Chapter 4, Chapter 6 and Chapter 5 of this thesis respectively.

3.1 Tshikondeni Coal Discard Dumps

Two co-disposal discard dumps exist at Tshikondeni Coal, the original dump which was established at commencement of the mine circa 1984, and the new dump which has been operational since early 2004. Both dumps were designed such that the flotation tailings are deposited in the centre section, whilst the coarse discard is used to build the outer walls. The two dumps are illustrated in Figure 3.1 and Figure 3.2, along with the grid pattern superimposed upon them for the intended auger sampling campaign described in the next section.

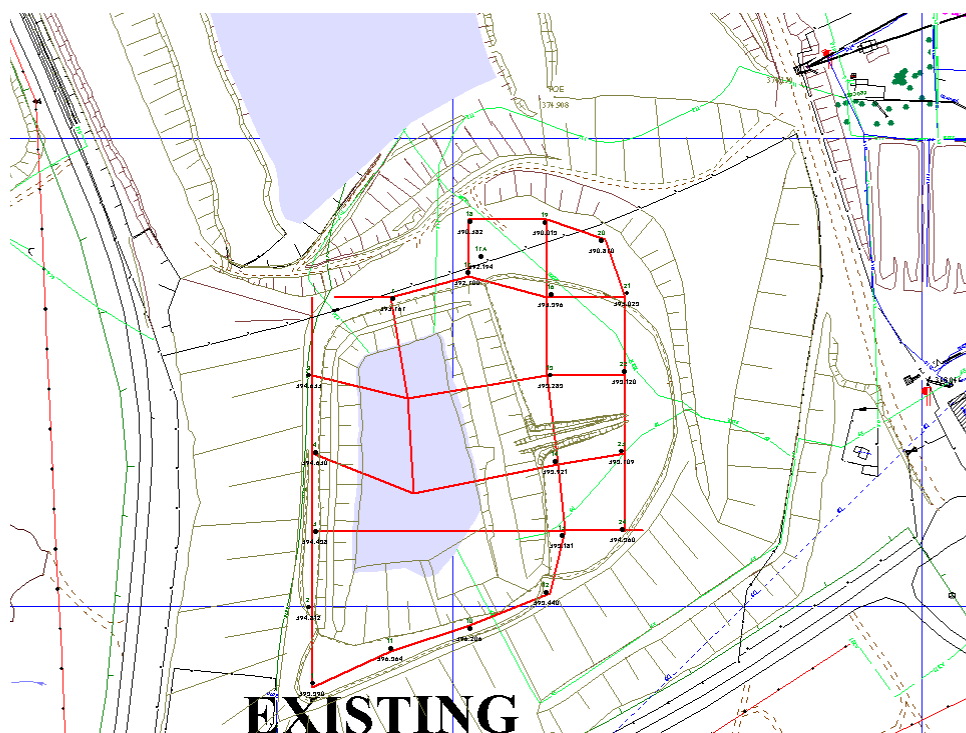


Figure 3.1: Original discard dump and planned borehole grid pattern

The original dump contains approximately 2.5Mt of discard at an ash content in the order of 50% for the coarser (+0.5mm) material and 30-35% for the fine (-0.5mm) material in the central section.

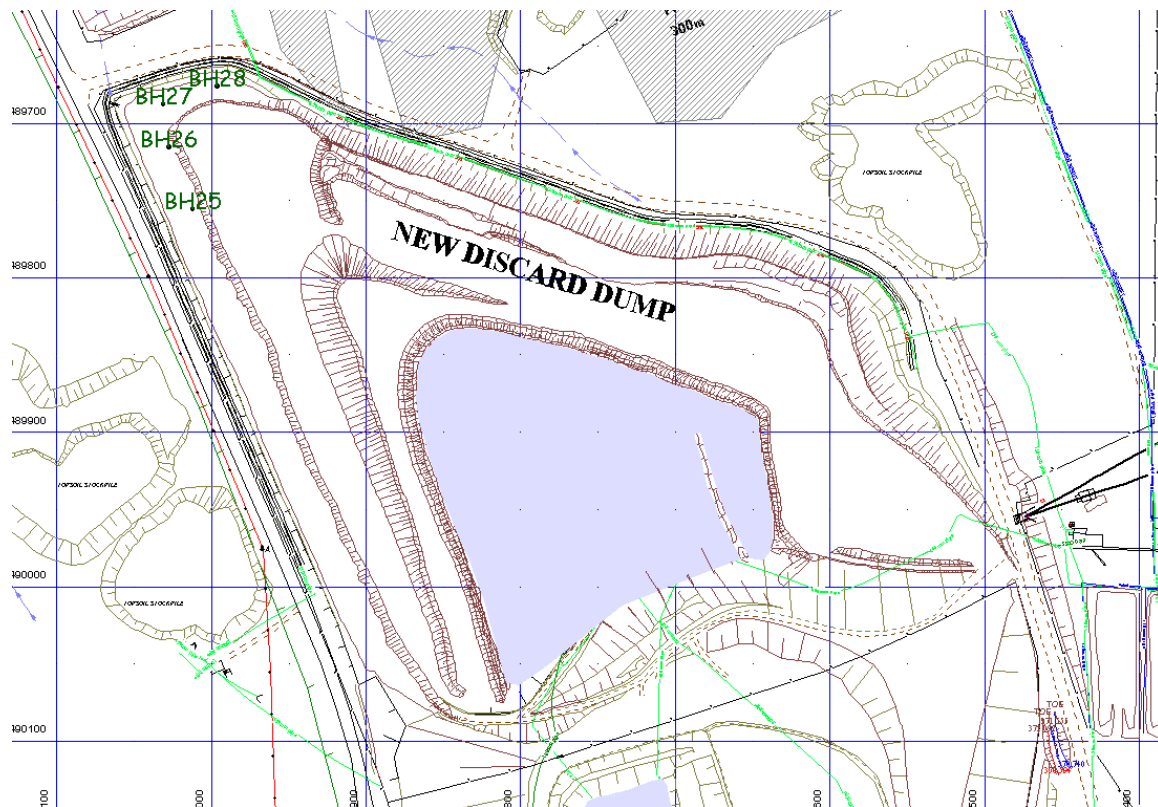


Figure 3.2: Position of boreholes on the current discard dump

The current, operational dump contains approximately 1.8Mt of material with a coarse ash in the order of 55%, whilst the centre section is similar to the old dump at 30-35% ash. The higher ash in the coarse discard being attributable to a change of product specification from 12.3% ash to 14.0% ash in 2004: the plant control being related to changes in the operating relative density of the dense medium cyclones.

An old return water dam on the property, known as the “Visgat dam”, was also added to the scope of the work. It contains very fine material from the early operating days of the plant and which would need to be removed and rehabilitated as part of the mine closure process. This project therefore provided a simultaneous opportunity to both recover material and remove an environmental liability. The dam layout is presented in Figure 3.3. The dam is approximately 450m long and 300m wide, has a surface area of approximately 5500m² and contains approximately 27 500 tonnes of fine material.

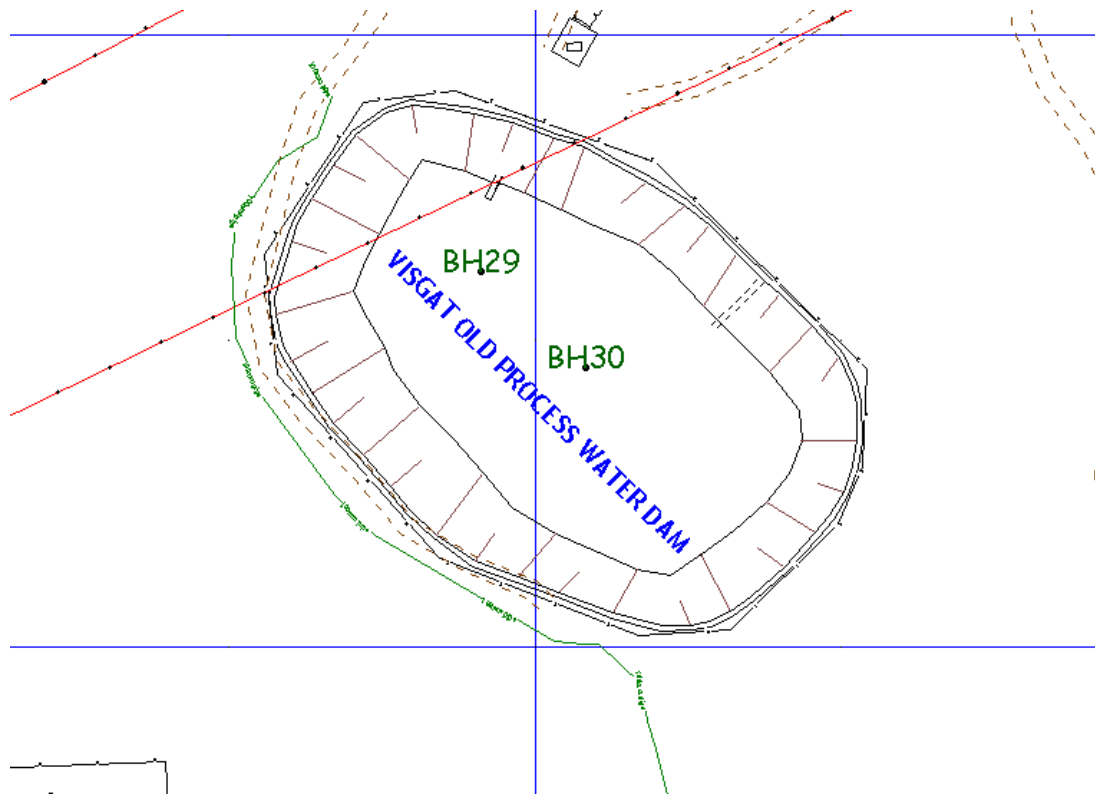


Figure 3.3: Position of boreholes from the Visgat dam

3.2 Auger Sampling Test Campaign of the Dumps

3.2.1 Auger Sampling Procedure

One of the important facets of this project was to characterize the discard material present on the original discard dump by means of auger drilling of borehole samples. A grid pattern was superimposed upon a plan of the dump to allow for holes to be drilled at a spacing of 50m between holes. The dump and the grid pattern are shown in Figure 3.1 in the previous section, which included a pair of holes drilled in the fines section since it had dried out.

The old dump had previously been rehabilitated and the topsoil was removed by a front end loader until the top of the coal was exposed. The auger drill design was such that only 1.5m core lengths could be extracted at a time and therefore three of

these lengths were used to form one sample. This was to allow for sufficient material to be available for the analytical work and would also provide a convenient height for mine planning, which typically works at bench heights in the order of 5m. The drilling of one of the holes on the original dump is shown Figure 3.4 to Figure 3.6, the central section containing the plant fines (flotation section tailings) is clearly seen.



Figure 3.4: Auger drill view 1



Figure 3.5: Auger drill view 2



Figure 3.6: Auger drill view 3

Whilst the drilling contractor was on site, an additional six holes were drilled to enable further information to be gathered. Four of these were taken from the current dump in an area not under construction at the time and away from mobile machinery working on the dump for safety reasons as indicated in Figure 3.2 in the previous section. These samples are acknowledged as not being fully representative of the material on the dump, however, several sources of discard being deposited upon this dump were also sampled as the project progressed as a means of mitigating this issue.

The final two were taken from the disused “Visgat” return water dam as depicted in Figure 3.3 in the previous section.

3.2.2 Sample Characterization Procedure

For each hole, the 1.5m long incremental core lengths, the maximum length the auger could sample, were combined to form composite samples of 4.5m long to represent what may be a typical “mining” bench height should the dump be found to be suitable for reclamation. The composite samples were then air dried, weighed, mixed well and split so that three quarters of the sample was available for testing and the remaining quarter stored as a reference sample for any future work.

Each 4.5m composite sample was screened at 0.5mm, the +0.5mm fraction was then destoned by subjecting it to a single density float and sinks step at a relative density of 1.80. The floats fraction was then combined with the -0.5mm fraction, milled to $-212\mu\text{m}$ and subjected to a flotation rate test. At each stage, a representative analytical sample was extracted, prepared and the ash content determined according to the relevant ISO standards.

This process is illustrated in Figure 3.7 overleaf.

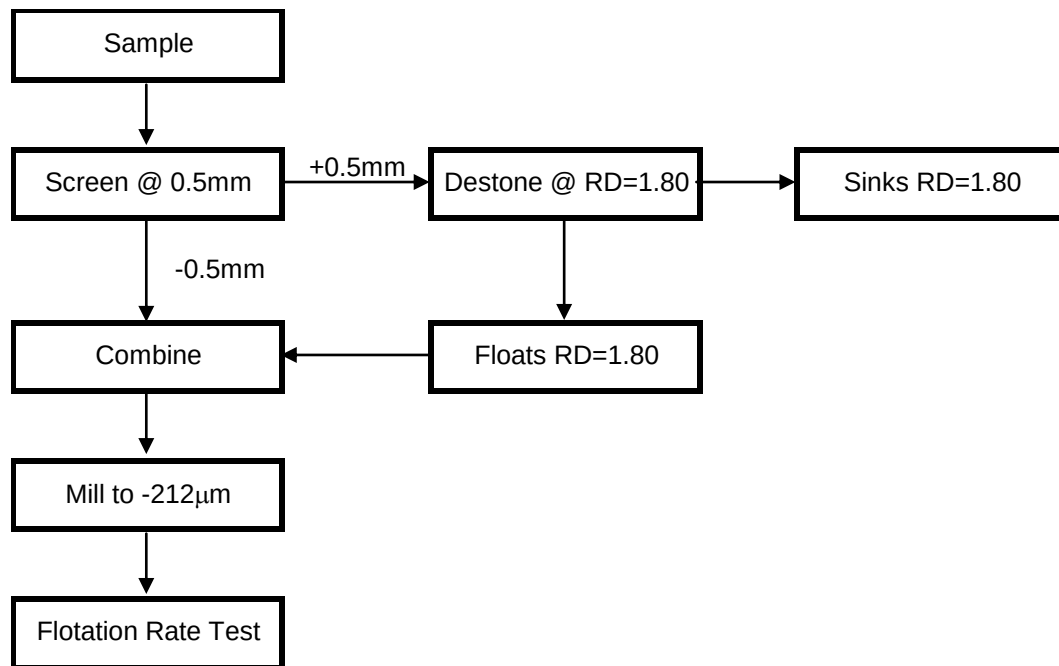


Figure 3.7: Borehole sample test procedure

3.2.3 Sample Characterization Results

The results to be discussed in this section will only be the raw “in situ” ash grades to develop an assessment of the overall quality of the old and new dumps.

The work related to destoning and froth flotation will be discussed in detail under the relevant sections along with the other samples considered in this project. This includes the two Visgat samples, which were found to be -0.5mm in size terms and are discussed in the flotation section.

3.2.3.1 Assessment of the old dump

The detailed results are presented in full in Appendix 2. The revised drilling sequence is illustrated in Figure 3.8, which is slightly different to that presented earlier in Figure 3.1 above (in that borehole BH6 could not be drilled and was replaced by an additional sample named BH17a) as shown on the plan. Samples BH8 and BH9 were extracted from the slimes portion of the dump which is illustrated in blue on the diagram.

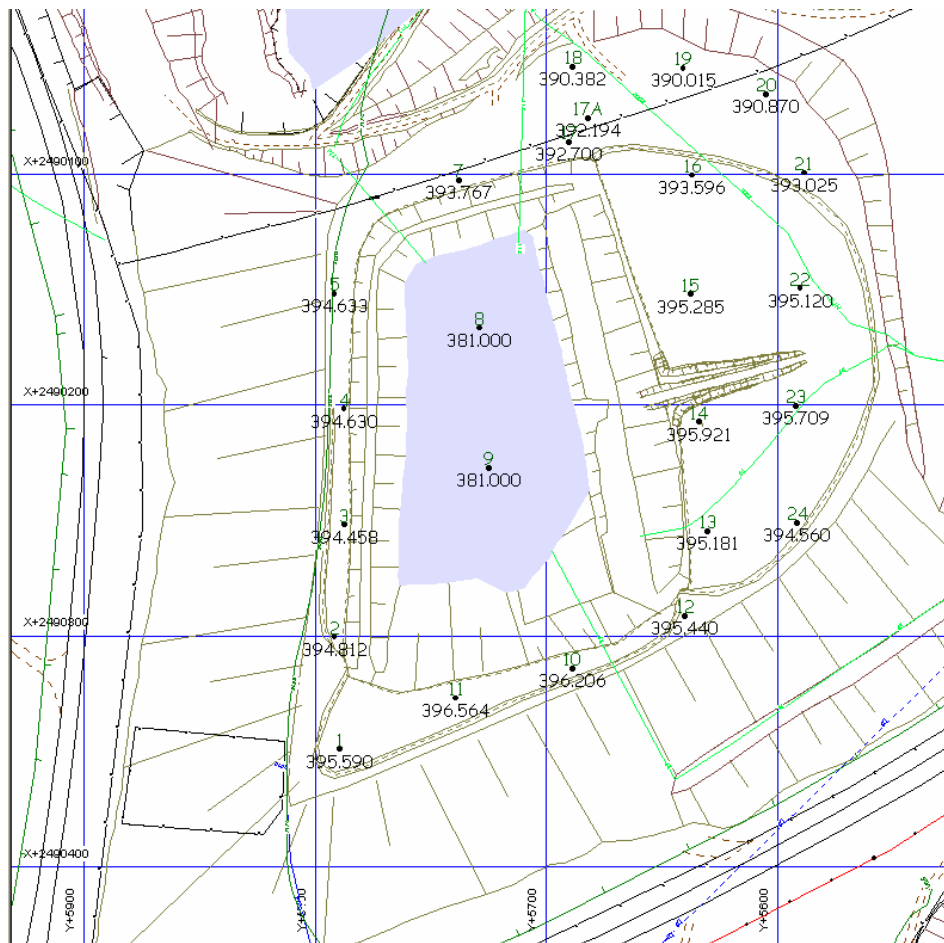


Figure 3.8: Revised drilling pattern for the old discard dump

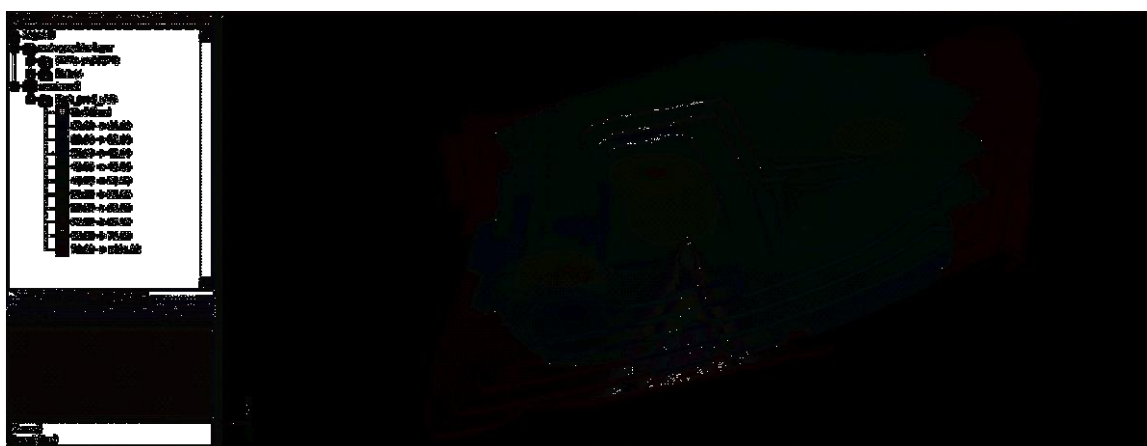
Despite the dump being around 30m high at the highest point, a layer of hard rock had been previously deposited earlier in its development and therefore the auger was unable to drill through this and reach the bottom of the dump. The maximum length of core obtained was 22.5 m from borehole samples BH3 and BH4 respectively.

A statistical examination of the ashes obtained from each borehole and the respective depths below the dump surface, except for BH8 and BH9 taken from the slimes dam section of the dump, are presented in Table 3.1, the first column representing all results obtained and the subsequent columns according to depth below the surface of the dump.

Table 3.1: Descriptive statistics of ash content of dump samples

	All Results	Depth Below Surface Of Dump				
		0-4.5m	4.5-9.0m	9.0-13.5m	13.5-18.0m	18.0-22.5m
Mean	50.3	51.3	49.8	49.8	50.6	48.8
Standard Error	0.3	0.7	0.4	0.5	0.6	4.2
Median	50.0	50.8	49.2	49.7	50.5	53.0
Standard Deviation	2.8	3.3	1.8	2.2	2.9	7.3
Sample Variance	8.0	11.0	3.1	4.8	8.3	53.3
Kurtosis	4.3	3.8	0.8	6.1	0.7	-
Skewness	0.4	1.8	1.2	-1.7	0.2	-1.7
Range	21.3	14.0	6.4	10.8	13.0	12.7
Minimum	40.4	47.7	47.7	42.3	44.5	40.4
Maximum	61.6	61.6	54.1	53.1	57.5	53.0
Count	90	22	22	22	21	3
Largest	61.6	61.6	54.1	53.1	57.5	53.0
Smallest	40.4	47.7	47.7	42.3	44.5	40.4

It can be seen that the ash content of the dump in global terms was in the order of 50%, though a significant range of values between lowest to highest at 40.4% and 61.6% respectively were noted. Examination of the various layers below the depth of the surface of the dump was also noted to produce similar results. Figure 3.9 to Figure 3.13 show a three dimensional and two dimensional plan view representations of the dump by layer according to raw ash contents from the borehole data.

**Figure 3.9: 3D View of dump showing ash distribution based on borehole data**

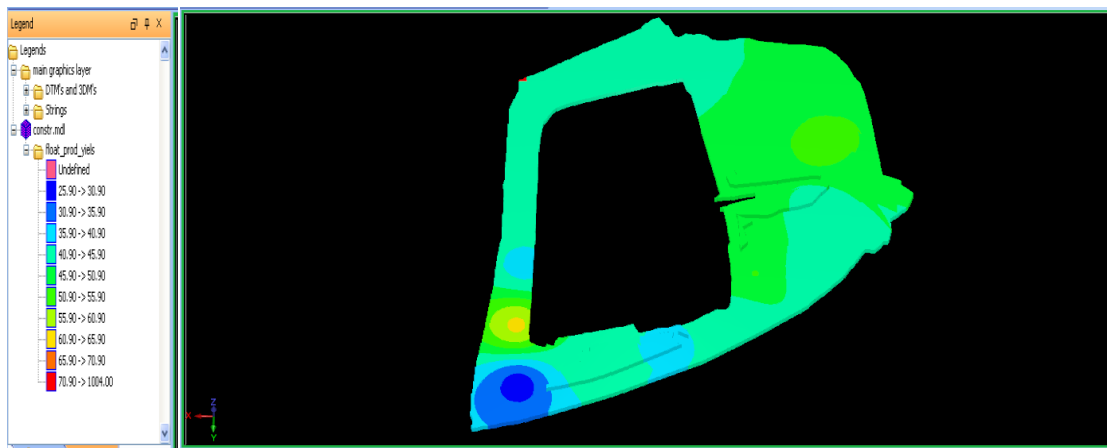


Figure 3.10: Plan view of dump showing ash distribution based on borehole data 4.5 to 9.0m below dump surface

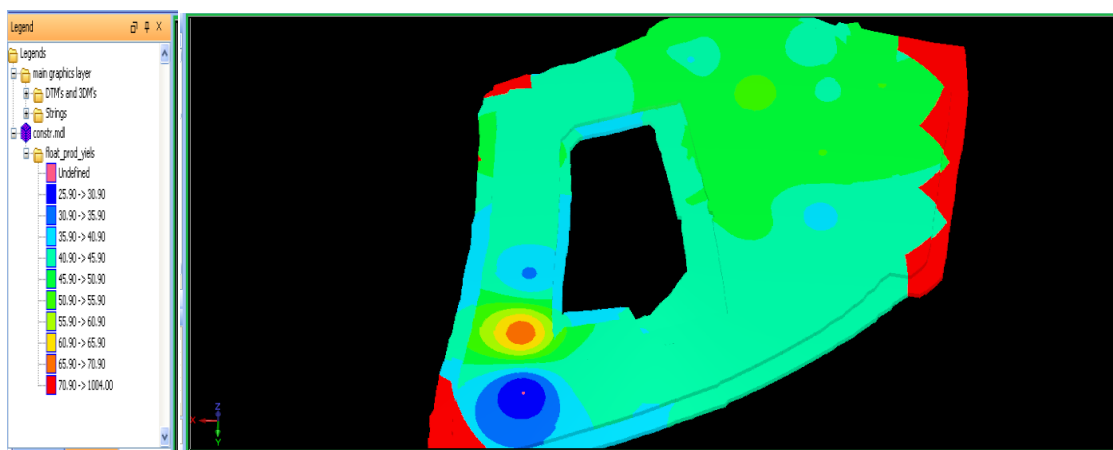


Figure 3.11: Plan view of dump showing ash distribution based on borehole data 9.0m to 13.5m below dump surface

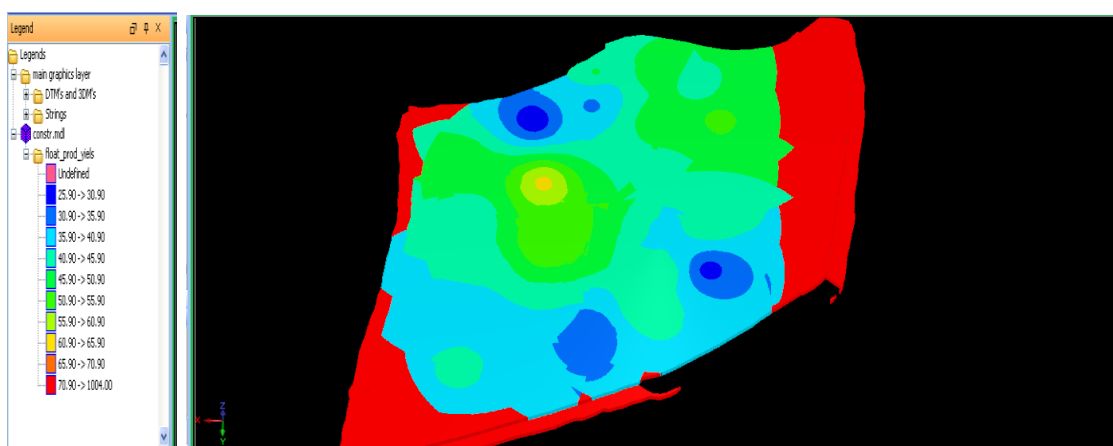


Figure 3.12: Plan view of dump showing ash distribution based on borehole data 13.5m to 17.0m below dump surface



Figure 3.13: Plan view of dump showing ash distribution based on borehole data 17.0m to 22.5m below dump surface

3.2.3.2 Assessment of the new dump

The detailed results are presented in full in Appendix 2. These samples were taken from only a small section of the dump by the drilling team due to safety considerations around human interaction with moving machinery, and therefore the results obtained may not be fully representative of the entire dump. A statistical examination of the ashes obtained from each borehole and the respective depths below the dump surface are presented in Table 3.2.

Table 3.2: Descriptive statistics of ash content of dump samples

	All Results	Depth Below Surface Of Dump		
		0-4.5m	4.5-9.0m	9.0-13.5m
Mean	55.8	56.4	54.8	56.6
Standard Error	0.5	0.8	0.9	1.2
Median	56.0	56.7	55.3	56.6
Standard Deviation	1.7	1.5	1.7	1.7
Sample Variance	2.8	2.3	2.9	2.9
Kurtosis	0.5	1.4	0.6	0.0
Skewness	-0.7	-0.8	-1.2	0.0
Range	5.6	3.6	3.6	2.4
Minimum	52.4	54.4	52.4	55.4
Maximum	58.0	58.0	56.0	57.8
Count	10	4	4	2
Largest	58.0	58.0	56.0	57.8
Smallest	52.4	54.4	52.4	55.4

It can be seen that the ash content of the dump in global terms was in the order of 56%, though compared to the new dump, a significantly smaller range of values between the lowest to the highest at 52.4% and 58.0% respectively were noted.

3.2.3.3 Assessment of the Visgat Dam

The material extracted from the cores in the Visgat Dam was noted to be below 0.5mm in all instances and had in-situ ash contents of 24.5%, which was higher than that of the fines sections of both the old and current dumps.

Only one core length of 4.5m for each of the two holes could be taken due to the shallow nature of the dump, which was to be expected. The samples could not be subjected to dense medium separation due to their fineness and were only subjected to froth flotation as discussed in Chapter 6 of the thesis.

3.3 Chapter Summary

A total of 30 auger borehole samples were extracted from the two discard dumps and the Visgat Dam, of which 24 samples came from the old dump, 4 samples from the current operational dump, and 2 from the Visgat Dam respectively.

The individual boreholes were divided into 4.5m long sub-samples to represent a typical mining bench and each subsample examined for the in-situ ash content, which was found to approximately 50% in the old dump, 55% in the current dump and 24.5% in the Visgat Dam.

A three dimensional model showing the ash distribution in the old dump was developed along with the graphical representations of the ash distribution through the dump in consecutive 4.5m benches.

The samples were then subjected to upgrading by destoning via dense medium separation as discussed in the next chapter.

Chapter 4: Destoning Of Coal Discards

This chapter examines in detail the various techniques available to destone i.e. remove high density waste rock from the discards produced by a coal preparation plant. Thereafter, the application of the most appropriate technologies for the Tshikondeni plant are examined.

It begins with a literature survey examining current and relevant methods of destoning using technologies that could be applied in the South African coal processing environment. The performance and suitability of dense medium separation process equipment, gravity separation equipment, and recent developments in screening technology were examined and the current or recent application on South African coal mines discussed.

Testwork was undertaken on auger core samples from the discard dumps, bulk samples from the various operating shafts, as well as a bulk sample from the old discards dump to examine the effect of destoning using:

- Screening only
- Dense medium separation
- Gravity separation using an air jig, which was found not to be suitable for Tshikondeni material
- Simulations were undertaken for a two stage process using screening to perform the initial destoning stage by removing oversized high ash material ahead of dense medium separation for the smaller size fractions.
- The initial simulations for the production of a hard coking coal product similar to the current plant specification produced low yields and therefore further simulations were performed to see if a product suitable for supply to the national power producer (Eskom) was viable. This was also noted to be too low to be economically feasible given the remote location of Tshikondeni relative to the power stations in South Africa

4.1 Literature Survey

The discards from any coal processing plant contain essentially three components:

- i. A coal fraction that was rejected in the process mainly due to it not making the specification required by the plant,
- ii. Misplacement of saleable material into the discard due to inefficiencies of the washery units, and
- iii. Waste rock such as shales and sandstones.

It is therefore necessary to define where the discard changes from “coal” to “stone”. Horsfall (1987) and Horsfall (1992) indicated that it is generally accepted in the South African coal processing industry, that material with a specific gravity of less than 1.80g/cm^3 would constitute “coal”, whilst material with a specific gravity greater than this would constitute “stone”.

Destoning processes are relatively new in South Africa and current practice uses both dense medium separation and water based jigging. These options are described in greater detail below, along with dry jigging, a technique which has been tested and could be applied within South Africa, where water can be considered a scarce resource.

4.1.1 Larcodems

Woodman et al (s.a) reported that the Larcodems (an acronym for LArge Coal Dense Medium Separator) was developed in the United Kingdom at the then National Coal Board's Technical Services Research Establishment at Bretby in the early 1980's. It was conceived as a low cost, efficient alternative to traditional Baum jigs, dense medium drum and dense medium cyclones. The original installation at Point Of Ayr Colliery treated a feed size of $100 \times 0.5\text{mm}$.

The Larcodems is illustrated in Figure 4.1, where it is noticed that the unit is similar to a Dyna-Whirlpool (illustrated in Figure 4.2 for comparison), but with the addition of

a vortex extractor to facilitate the extraction of medium and discard material. The medium is introduced into the unit at the product end, whilst the feed enters from discard end. Like many cyclonic separators, the vessel typically operates at a medium pressure equivalent to nine times the unit diameter.

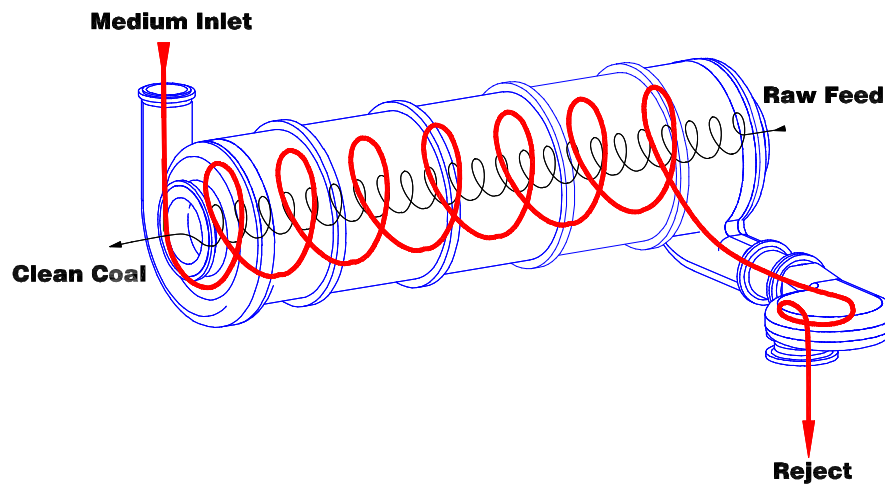


Figure 4.1: Larcodems (Woodman et al, s.a)

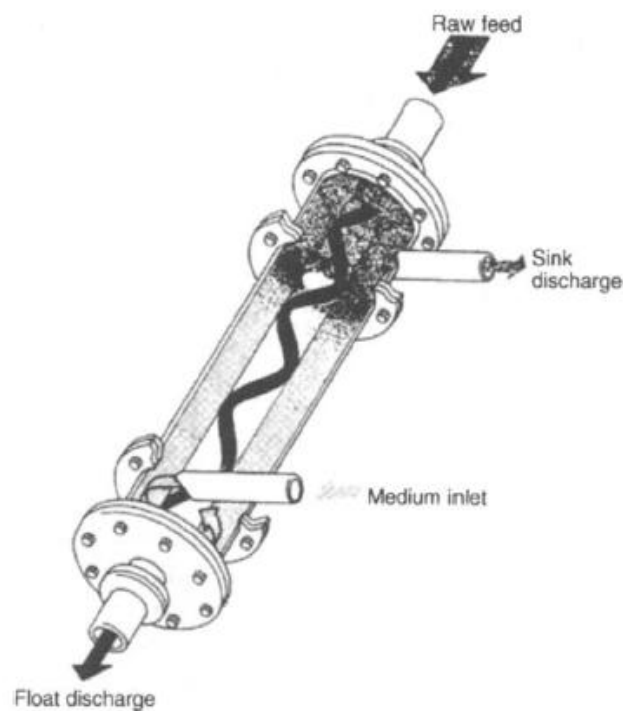


Figure 4.2: Dyna-Whirlpool Separator (Wills and Napier-Munn, 2006)

Harrison et al (1998) reviewed the performance of the unit in South Africa, where it was first used outside of the UK. The first of these installations was for Randcoal's (now BECSA's) Middelburg Mine Services South Section treating 350tph of -100+20mm or -100+12mm raw coal through a 1200mm vessel in a high gravity i.e. destoning application. The bottom size being dictated by the raw coal quality. The second South African installation, albeit not in a destoning role, was at Xstrata's Tavistock No.4 Seam Plant, and was used to process -80+10mm material at a RD of 1.65 to produce a 26.8MJ/kg product.

The Middleburg Mine Services installation was subject to an independent acceptance efficiency test as detailed by Shehab (1995). The test results are summarized in Table 4.1 and the partition curve for Module 1 depicted in Figure 4.3. From the results, it was noted that the vessel was able to operate in an efficient manner in a destoning application. Of interest, is that the ecart probable measurements also included the 90/10 and 95/5 sections of the curve in addition to the traditional 75/25 approach to enable an estimated of the effect of misplacement of material to the floats and sinks to be better quantified. In terms of these, Module 1 was noted to be the more efficient

Table 4.1: Summary of plant efficiency test results (Shehab, 1995)

	Module 1	Module 2
RD Cut Point (d_{50})	1.8106	1.8776
Ep	0.0051	0.0108
Ep 90/10	0.0305	0.0609
Ep 95/5	0.0600	0.1080
Organic Efficiency	99.89	99.86

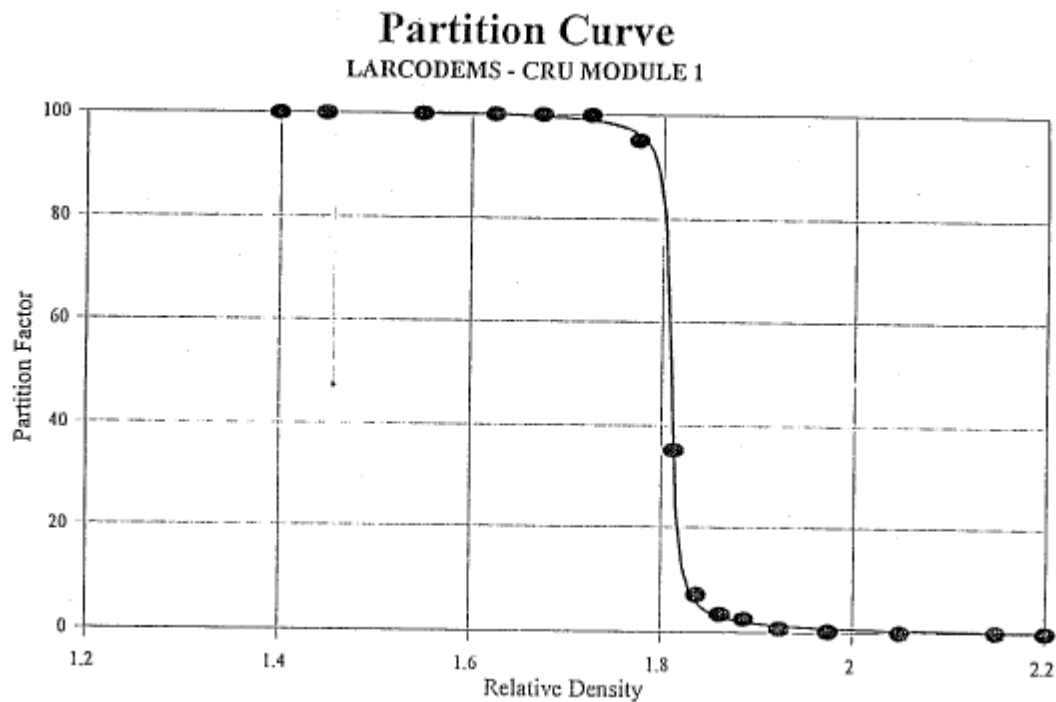


Figure 4.3: Module 1 partition curve (Shehab, 1995)

4.1.2 Tri-Flo Separator

The Tri-FloTM separator is a multi-stage dense medium which Bozzato (2007) states is an ideal technology for discard re-washing due to the high sinks capacity, and can produce three products (a product, middling and discard) in a single unit. This is well illustrated in Figure 4.4 where it is clearly seen that the material is fed into the vessel separately from the medium. The unit has the visual appearance of being two Dyna-Whirlpool separators joined together.

The first stage float material directly enters the second stage for a second separation. In difficult two-product separations, only one circulating medium is used, and the second stage further washes the float material for improved sharpness of separation. In a three product unit, the second stage would use a lower gravity cut relative to the first stage.

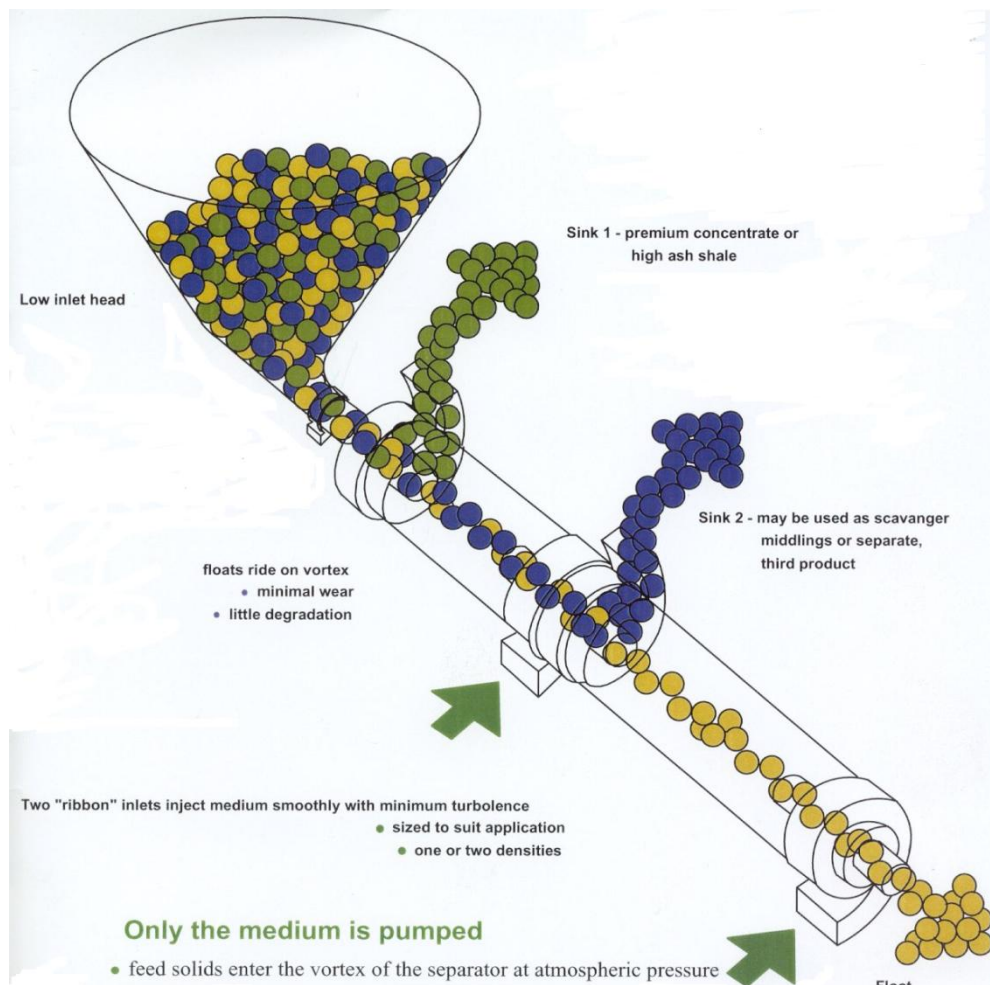


Figure 4.4: Schematic of a Tri-Flo™ Separator (Jacobs, 2007)

Bozzato (2007) summarized preliminary work on a 700mm internal diameter vessel on coal discards in Table 4.2, which considered various size fractions and separating RDs of 1.85 and 1.90 respectively. The vessel was operated as a rougher-cleaner.

Table 4.2: Results on washing discards in a 700mm Tri-Flo™ (Bozzato, 2007)

Relative Density	Test Result	Size Fraction (mm)			
		50 x 25	25 x 6	6 x 0.5	Average
1.85	Feed Ash (%)	54.6	46.2	37.3	46.4
	Product Ash (%)	31.8	29.0	25.6	28.4
	Ep	0.010	0.015	0.019	0.017
1.90	Feed Ash (%)	61.1	49.6	42.2	51
	Product Ash (%)	32.5	30.8	27.1	29.9
	Ep	0.015	0.018	0.014	0.016

The results indicated that low epm values where possible and that the feed ash could be reduced significantly, from the order of 50% ash to just under 30% ash. No

indication of organic efficiency or misplacement of material was reported, though these were reported on the same tests by Jacobs (2007) to be 84.3%, 95.7% and 96.0% for the 50x25mm, 25xmm and 6x0.5mm fractions respectively at a relative density of 1.85; and 83.8%, 99.4% and 96.5% for the three respective size fractions at a relative density of 1.90

Bozzato (2007) further reported that a full scale 700mm internal diameter unit, treating 250tph plant feed has been erected in South Africa and is being used to process either discards or low quality ROM sources. The plant can cope with up to 210 tph of discards without sacrificing product quality as suggested in Table 4.3, where the clean coal yield calculated in heat terms fell from 90.7% to 16.7% when treating a discard source.

Table 4.3: Tri-Flo™ product quality in 2-product configuration (Bozzato, 2007)

Feed Type: Discard	Feed		Clean Coal		Rejects		Clean Coal Yield (% on CV Basis)
	Ash (%)	CV (MJ/kg)	Ash (%)	CV (MJ/kg)	Ash (%)	CV (MJ/kg)	
Low Ash	32.3	19.45	27.8	20.83	65.5	5.97	90.7
High Ash	63.1	5.81	30.4	20.18	69.6	2.94	16.6

The results presented in Table 4.4 show the unit being operated on run of mine as a 3-product separator producing an export quality product, a middling product suitable for local power product and a discard.

Table 4.4: Tri-Flo™ product quality in 3-product configuration (Bozzato, 2007)

Feed Type: ROM	Feed		Clean Coal		Middlings		Discards	
	Ash (%)	CV (MJ/kg)	Ash (%)	CV (MJ/kg)	Ash (%)	CV (MJ/kg)	Ash (%)	CV (MJ/kg)
RD1=1.82/RD2=1.61	19.5	24.99	19.0	25.21	38.5	16.63	59.7	7.31
RD1=1.70/RD2=1.49	25.8	22.21	14.1	27.36	24.4	22.83	53.0	10.24
RD1=1.66/RD2=1.40	18.3	25.52	13.2	27.75	21.9	23.93	53.1	10.21

No yield data nor process efficiency information were given for the full plant trials, which limited a more detailed assessment of the vessel.

4.1.3 Jig

Jigs are widely used in coal processing operations throughout the world, though no new installations have been erected in South Africa for several decades due to the industry favouring dense medium separation, and the last operational unit was decommissioned in May 2000 (Dieudonne et al, 2002).

Jigs offer several advantages in terms of high density separation in that they only require air and electricity as the main consumables. Water is recirculated in the process and the clarity and level of fine coal contamination is less critical than for a dense medium process. A jig can also handle a wide size range of feed material (0x150mm).

Difficulties with jigs are the failure to accommodate the higher proportion of near gravity material found in South African coals. With coals that are of low to medium “difficulty” to wash, i.e. up to 15-20% near density material, a jig can be an effective and the cheapest method to beneficiate coal (Dieudonne et al 2002, Sanders et al 2002a).

Excessively strong pulsing results in discard being misplaced to the top, clean layers; whilst weak pulsing results in product being lost to the discard stream. The pulse wave form has been found to be an important aspect in the operation of the jig (Dieudonne et al 2002, Loveday and Jonkers 2002) and a typical example of the pulse with respect to time is illustrated in Figure 4.5 (Claasen and Lundt, 2007). The effect of the pulse cycle is shown for both the water (thin line) and the solids (bold line) across four time periods: T1 is the pulse time, T2 is the hold time, T3 is the exhaust time, and T4 is the delay time. T2 is required to keep the water in a top hold cycle to allow the heavier particles to sink through the lighter coal layer whilst T4 is required to balance the cycle time. The separation of coal from discard is clearly seen under T3, where the green coal particles sit on top of the black discard layer.

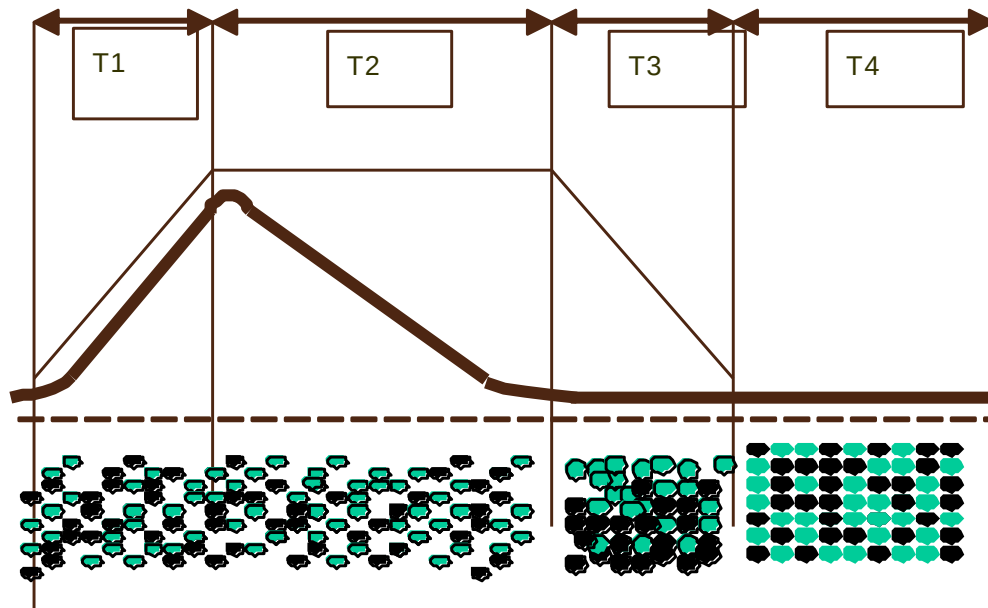


Figure 4.5: Jig pulse wave form (Claasen and Lundt, 2007)

Loveday and Jonkers (2002) reported performance data on a jig for a variety of coals for finer sizes, similar to the Tshikondeni plant feed size, which has been summarized in Table 4.5. In all cases a significant reduction in the ash was observed, though yield data was not presented. However, it is clear that a jig can be an effective method of destoning a source of coal.

Table 4.5: Fine coal measured performance for various coal sources (Loveday and Jonkers 2002)

	Feed		Product			
	Ash (%)	t/h/m	Ash (%)	d ₅₀	epm	Imperfection
2-13mm Australian Coal	40	80	25	1.95	0.09	0.09
0.5-2mm Australian Coal	-	-	-	2.15	0.19	0.17
0.3-8mm USA Coal	13	75	7	1.7	0.08	0.114
0.5-16mm German Coal	30.6	63	6.8	1.6	0.07	0.117

4.1.4 ROMJIG

The ROMJIG was developed by KHD Humboldt Wedag in 1984 and is different to other jigs in that the feed passes across a movable inclined screen plate. The jiggling action is created by hydraulically lifting the feed end of the screen and allowing it to drop back under at a rate of 38-43 times per minute. Under the jiggling

action, the material separates with the heavy particles next to the screen and the lighter particles remaining on top. The unit is only available in one size with a 2m bed width and feed rate of 350tph treating material in a nominal size range of -350+40mm. (Sanders et al, 2002a). A cross sectional view of the ROMJIG is illustrated in Figure 4.6.

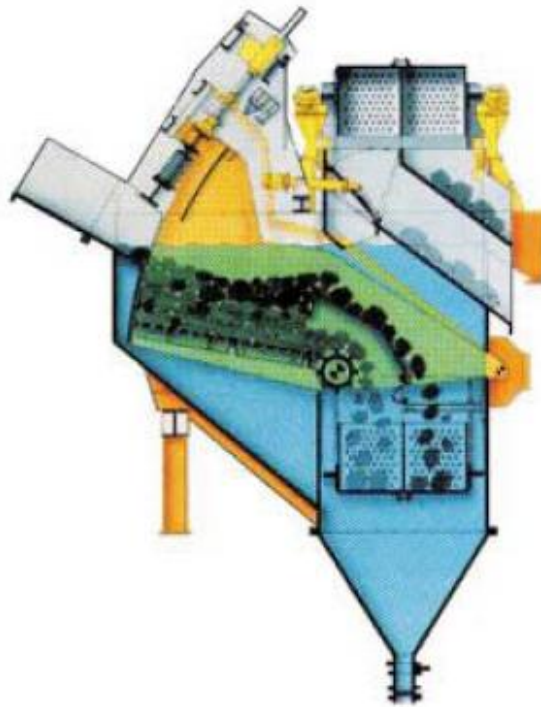


Figure 4.6: ROMJIG (Sanders et al, 2002a)

Sanders et al (2002b) published performance data from plants operating in India and China, which is summarized in Table 4.6. They also recommend that when the unit is being considered for a specific duty, that the coal and discard should be distinctly different, but that the amount of near density material is not too critical should the product be subjected to further treatment. However, if being used to produce a final product, it is preferable that little or no material should be present between relative densities of 1.70-2.00.

Table 4.6: ROMJIG performance data(Sanders et al, 2002b)

Plant	d₅₀	epm	Imperfection
Bina (India)	2.00	0.10	0.10
Bina	1.95	0.10	0.105
Bina	1.95	0.07	0.074
Kargali (India)	1.905	0.0825	0.091
Kargali	1.87	0.095	0.11
Fushun (China)	1.77	0.0705	0.092

Sanders et al (2002a) concluded that the ROMJIG is an effective method of cleaning large (>40mm) coals, has the lowest capital and operating costs compared to other process equipment treating the same size range. It can reduce the overall coal processing costs and increase plant capacity by removing contamination prior to entering the main plant. Furthermore, it can be installed in remote locations since it requires little water and no associated water treatment facilities.

One installation exists in South Africa and is used to remove stone prior to the product being conveyed several kilometres to the main processing plant. No data is available regarding the process performance of this unit. Given the size of the material in the Tshikondeni discard being less than 20mm, the application of the ROMJIG would probably not be suitable in this instance (Kottman, 2007).

4.1.5 Dry Pneumatic Processing in the U.S.A

Honaker (2007) indicated that dry coal cleaning was popular in the USA over the period 1930 to 1990 and that peak annual production was 25.4MT in 1965. The FMC Separator was a typical unit used circa 1940 and is depicted in Figure 4.7. The decline of the technology was due to the need for efficient low density separations allied with environmental and health concerns.

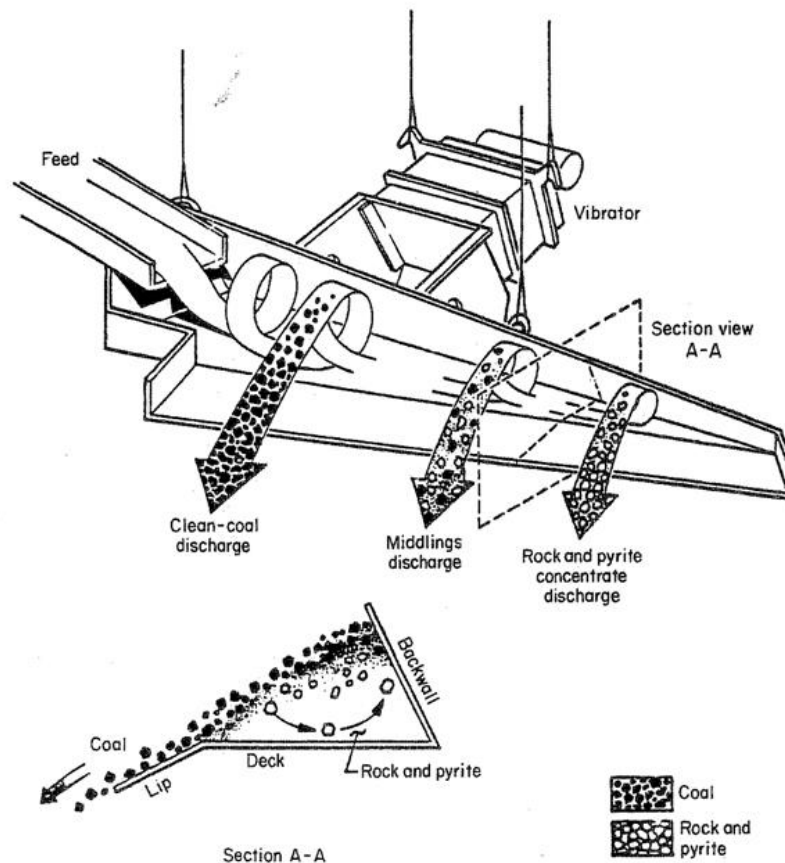


Figure 4.7: FMC Separator (Honaker, 2007a)

The recent resurgence in interest in dry separation is largely attributable to the need to reduce transportation costs and clean coals from the western regions. Dry coal has been demonstrated to effectively achieve density separations at relative densities above 1.85. Of recent interest is the Chinese developed FGX Separator illustrated in Figure 4.8, which is based on a riffing table utilizing air as the separation medium. It is a simple method of removing stone from raw coal and colliery discards, does not require the use of water, which is a scarce commodity in many parts of the world, and does not require large slurry ponds. Dust is problematic, however, and the discard still needs to be disposed of, which could lead to spontaneous combustion if not handled correctly.

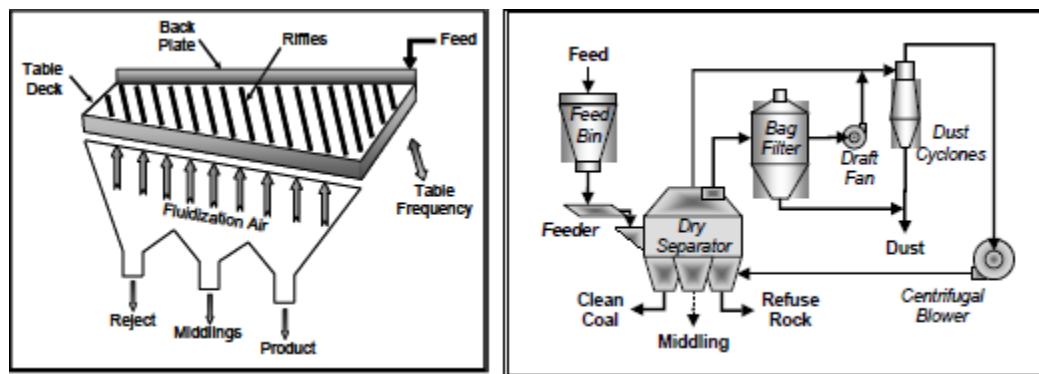


Figure 4.8: FGX Separator (Honaker, 2007)

It is seen that the feed enters the table from the far right corner whilst the fluidization air is injected through holes in the table. The presence of between 10-20% fines (-6mm material) in the feed is required to produce a fluidized autogenous medium bed. The coal is transported to the front of the table and discharged on the right, whilst the higher density material remains in contact with the table. The higher density material is removed to the back and the left by means of the vibratory motion of the table as new feed is introduced into the unit. The separation of the clean coal from the middlings and tailings is clearly seen in Figure 4.9.

Honaker et al (2007) reported results for two tests on coarse discard material using a mobile 5tph pilot plant. Washability data from the coarse (+6mm fraction) of the discard from an eastern Kentucky facility revealed that 45.3% of the material had a relative density of less than 1.80 with a heating value of 24.42MJ/kg, compared to the overall heating value of the discard at 12.58MJ/kg and ash content of 60.3%. Tests conducted indicated that the longitudinal slope of the table should be low and that a clean coal product yield of 44.5% at a heating value of 23.26MJ/kg was possible. The tailings qualities were noted to contain a heating value of 4.30MJ/kg and 81.5% ash respectively.

The second test was conducted at a facility in Virginia on a feed containing approximately 78% plus 6mm and an ash content of 55.5%. The amount of material floating at a relative density of 1.80 in the plus 6mm fraction was found to be 46.4%. Testwork showed that a product of 31.8% ash at a yield of 44.4% could be achieved from the plus 6mm fraction.

Typical mean probable error values in the order of 0.250 were reported and could be reduced to the order of 0.170 by recycling the middlings stream as depicted in Figure 4.10. The separations were noted to occur at relative densities of between 1.80 to 2.20.



Figure 4.9: FGX Separator distribution of products (Honaker et al, 2007)

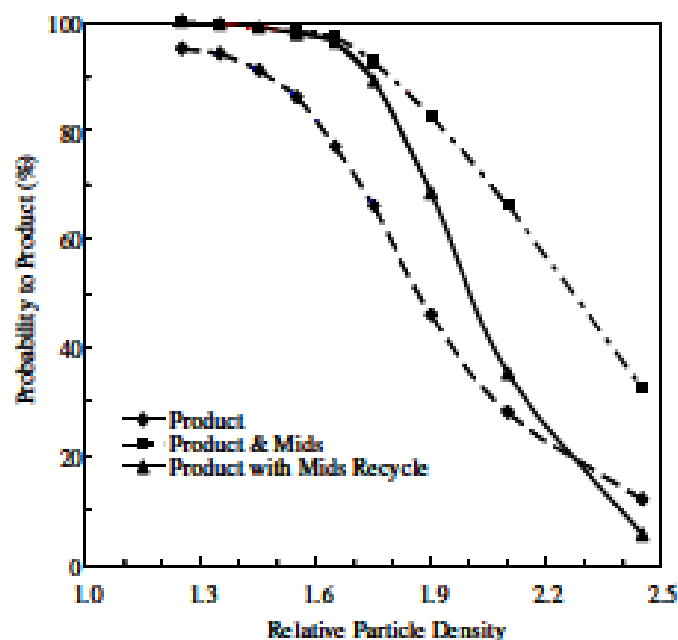


Figure 4.10: Partition curves showing FGX Separator performance (Honaker et al, 2007)

4.1.6 Dry Pneumatic Processing in South Africa

Exxaro Resources (PTY) LTD purchased a FGX Separator during 2008 which was tested on a variety of coal sources as reported by De Korte (2009). The unit is illustrated in Figure 4.11, whilst the results for the coarse coal are summarized in Table 4.7 and the normalized partition data depicted in Figure 4.12.

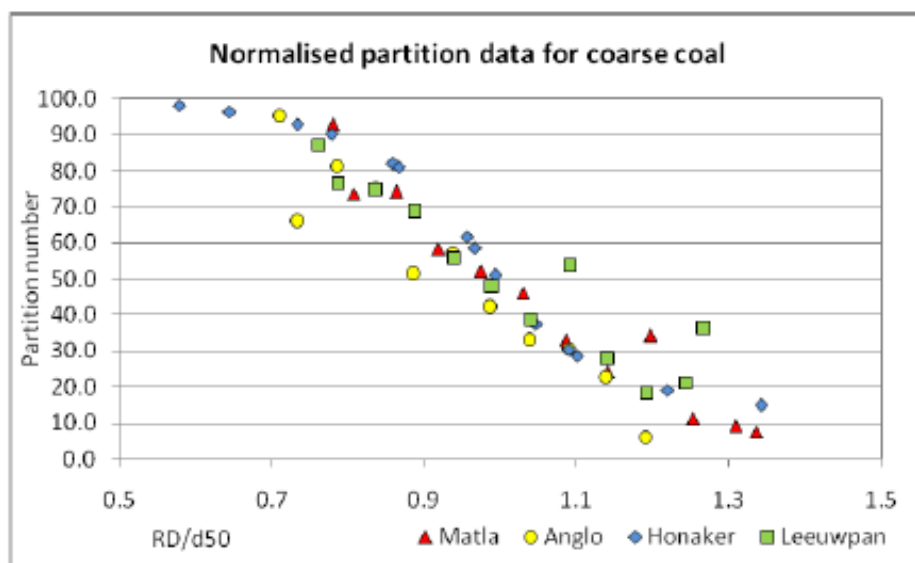
As indicated in the previously described work by Honaker (op cit.), the unit produced relatively high cutpoints with RDs of between 1.80 and 2.00, the epm values were in the order of 0.3 along with high proportions of misplaced material, though the amount of near density material was generally low. The normalized partition data in Figure 4.12 generally compares well with that observed by Honaker on USA coals.



Figure 4.11: Pilot Plant FGX Separator purchased by Exxaro Resources

Table 4.7: Summary of coarse coal results (De Korte, 2009)

Coal Type	d50	Ep	Total Misplaced (%)	Near Density Material (%)	Organic Efficiency (%)
Matla	1.797	0.267	24.9	7.4	68.6
Sasol	1.729	0.404	36.4	14.7	59.0
Anglo	1.975	0.257	27.1	20.0	71.1
Leeuwpan	1.972	0.307	28.1	6.5	76.5
Average	1.868	0.309	29.1	12.2	68.8

**Figure 4.12: Normalized partition data for coarse coal (De Korte, 2009)**

The minus 6mm size fraction was noted to be separated at higher relative densities than that of the coarse coal and it appeared that most of it reported to the product stream with very little upgrading. Figure 4.13 illustrates the fine coal partition data for several tests alongside results generated by Honaker for comparison, where by extrapolation, it is seen that very high separation densities in the order of 2.5 would be required.

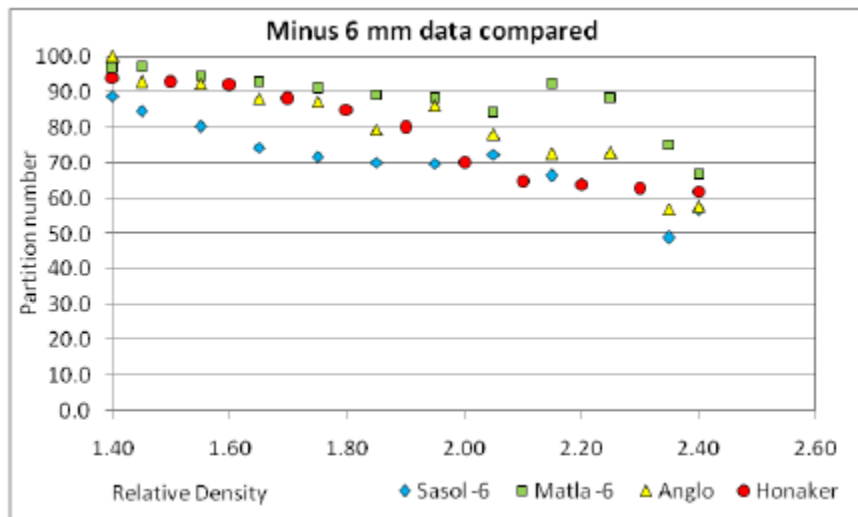


Figure 4.13: Partition data for minus 6mm coal (De Korte, 2009)

4.1.7 Destoning By Screening

Horsfall (1992) states that the level of cleaning of the run of mine coal is related to the requirements of the consumer and that the cleaner, i.e. lower ash content, the more complex the process would need to be. He presents five different “levels” of coal preparation ranging from basic crushing and screening, to processing of several size fractions and retreatment of discards to recover a middlings product. The most basic process, namely Level 1, is the screening out of the finer material and crushing of the oversize material, which is subsequently added back to the screened fines. No upgrading is achieved and the coal is merely crushed to a size suitable for the end user, the removal of the fines ahead of crushing applied to reduce fines generation and reduce the size of the installed crusher.

In terms of this study, a modification to Level 1 may be considered as an option in the destoning process for Tshikondeni discard, where the anticipated better quality finer material is removed by screening, and the anticipated poorer quality coarser material is discarded as depicted in Figure 4.14.

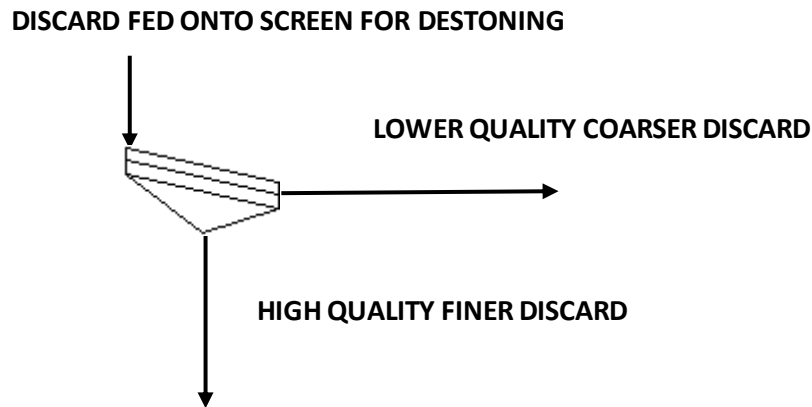


Figure 4.14: Destoning by screening only (after Horsfall, 1992)

In terms of this study the application of this step in the process flowsheet is dependent upon the results of a washability tests conducted on a range of suitable size fractions and will immediately give the more suitable of the following two options to follow:

1. The finer sizes contain recoverable values, the coarser size have potential to yield values by crushing prior to further processing, or
2. The finer sizes contain recoverable values, whilst the coarser material contains no values and crushing would not liberate any values from these sizes.

Lundt et al (2007) reported that screening at sizes finer than 6mm can be difficult due to clogging and blinding of the screen resulting in reduced efficiency. Screen efficiency is also influenced by the screen type and the moisture content of the material being screened. A screen using bi-vibration technology, the Bivi-TEC screen was tested at Leeuwpán colliery and was found to be a potential solution in that it could process damp fine coal without blinding, however the moisture content in the screen feed was found to negatively affect the overall screening efficiency.

The Bivi-TEC screen employs a unique dual vibratory screening process that eliminates clogging and blinding by using two weights vibrating at the same frequency, moving relative to each other and introducing a flapping motion on the flexible screen mats which alternately are tensioned and relaxed. The principle of

operation is shown in Figure 4.15, the screen mat in Figure 4.16 and the test unit installed at Leeuwpaan colliery in Figure 4.17.

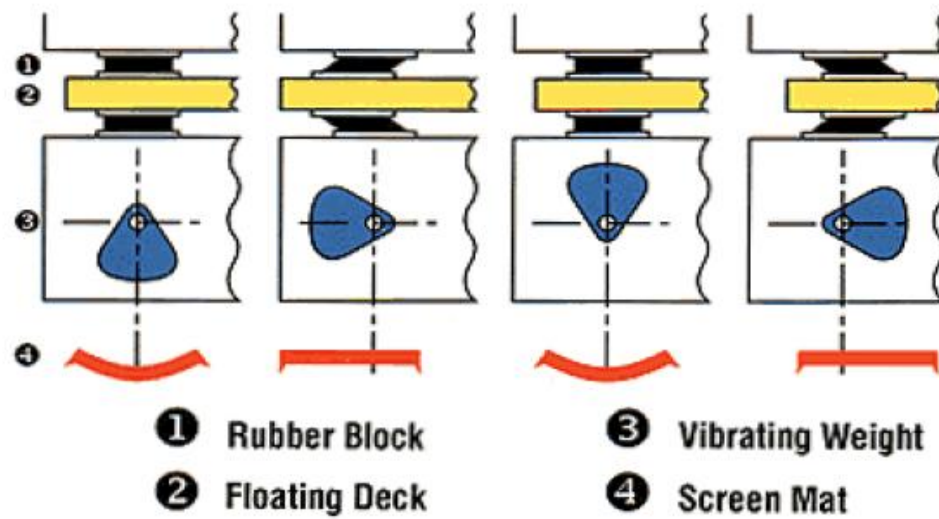


Figure 4.15: Bi-vibration motion of the Bivi-TEC screen (Lundt et al, 2007)

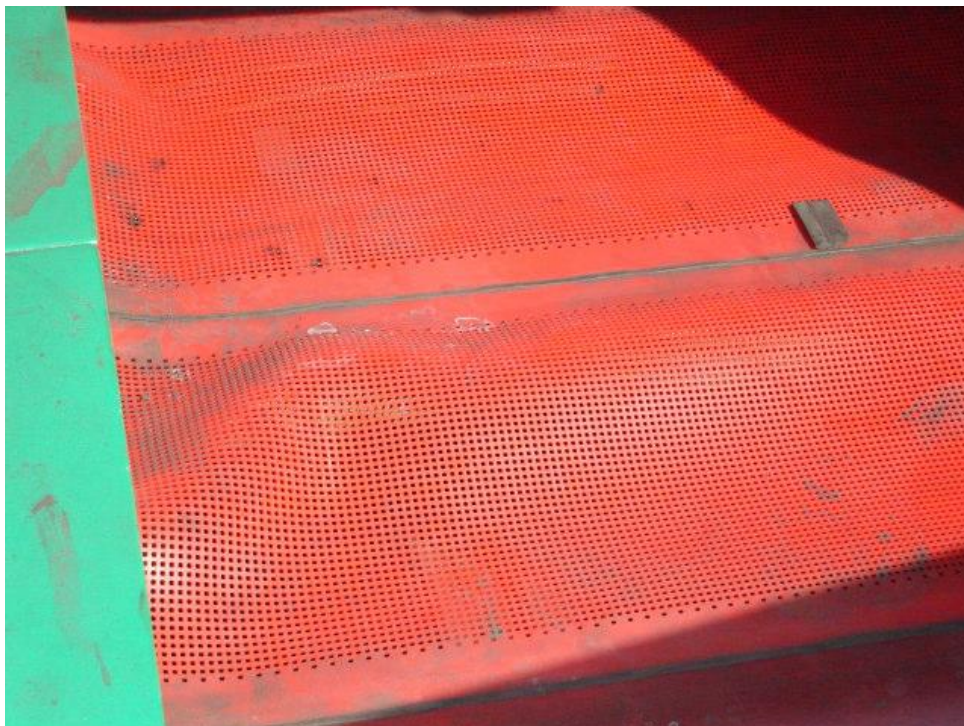


Figure 4.16: Bivi-TEC flexible screen mat (Lundt et al, 2007)



Figure 4.17: Bivi-TEC screen installation at Leeuwpan (Lundt et al, 2007)

Results obtained by Lundt et al (2007) from a series of tests as presented in Table 4.8, showed that the moisture in the feed controlled the screening process, i.e. as the moisture increased the amount of material reporting to the screen oversize increased, and the efficiency was reduced

Table 4.8: Summary of test results (Lundt et al, 2007)

Test Date	Test No.	Surface Moisture of Feed (%)	+3mm in Reconstituted Feed (%)	Feed Reporting to Oversize (%)	Screen Efficiency at 3mm (%)
09 May	1	6.6	46.3	32.9	87.9
21 May	2	8.3	26.2	92.5	30.1
30 May	3	12.7	21.3	98.3	9.4
05 Jun	4	N/A	26.6	85.3	48.3
05 Jun	5	7.9	27.1	66.4	75.7
13 June	6	5.8	24.2	88.1	43.5

Further work was undertaken on a laboratory scale using a 3.35mm screen to examine the effect of surface moisture as summarized in Table 4.9. It was noted that the screen efficiency, defined as material reporting to the screen undersize, also

dropped markedly above 9% surface moisture, and concurred to a large extent with the Bivi-TEC results presented in the previous table.

Table 4.9: Summary of laboratory tests (Lundt et al, 2007)

Feed Moisture (%)	Feed Reporting to Oversize (%)	Screening Efficiency (%)
0.49	26.24	97.2
1.66	42.19	92.7
4.40	45.19	91.5
9.41	56.65	85.5
7.02	49.78	88.9
11.02	83.94	58.5

Lundt et al (2007) concluded that the Bivi-TEC screen can handle fine material without any blinding or clogging issues, though surface moistures above 9% would result in a loss of efficiency. In a dry screen application such as the removal of fines from a dry feed such as the Tshikondeni discard dump, which is in an area where water is a scarce resource, this screen could have a high probability of success in the process flowsheet.

De Korte (2008) reported that the efficiency of the screening, as represented by the Ecart probable moyen (Ep or epm), was in the range of 0.200-0.250, and that efficient separations down to the order of 2.8mm were achievable.

4.1.8 Reflux Classifier

A fairly recent addition to the gravity separation stable is the Reflux Classifier, which falls within the latest generation of fluidized bed separators. Ghosh et al (2011) stated that teeter bed separation technology is commonly applied worldwide in coal preparation facilities to separate coal from waste rock in the -1.0+0.15mm size range, traditionally the size range that spirals would be used on in South African collieries. The advantage of the teeter bed units over spirals is that they can provide more efficient separations and higher throughput. The best separation efficiencies are noted to occur with narrow size ranges (3:1 top size to bottom size ratio being

reported as optimal), though commonly observed disadvantages are a sharp increase in separation density with a decrease in particle size and coal misplacement in sizes above 1mm.

As can be immediately seen in Figure 4.18, it is different to traditional teeter bed separators in that the top of the unit also utilizes lamella plates. These plates are used to increase the settling rate of the discard particles and also enhance throughput capacities.

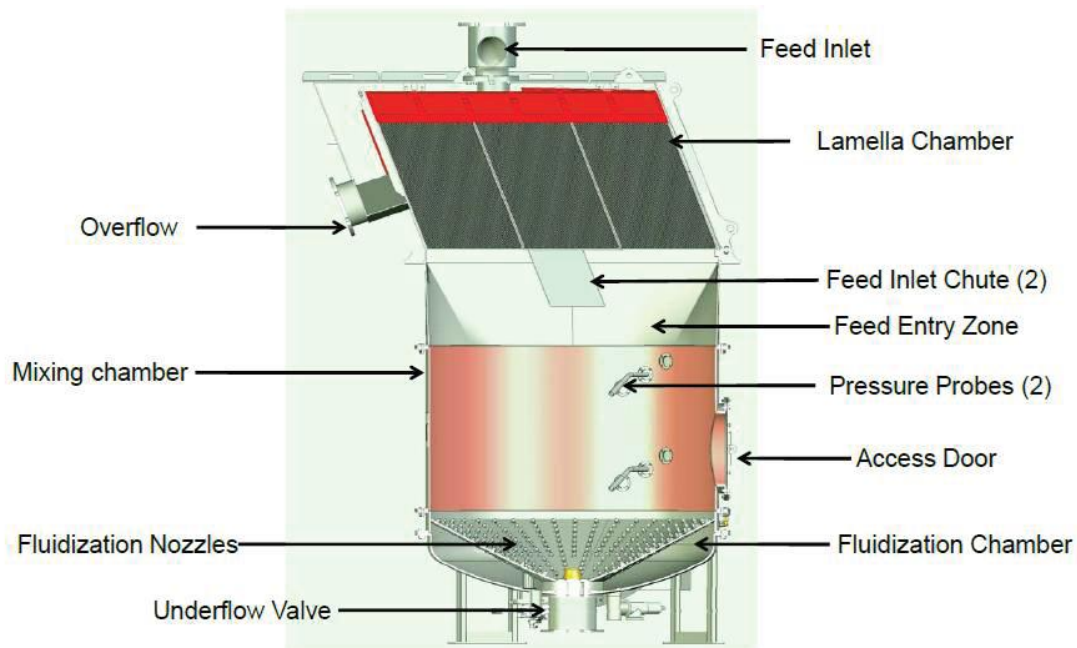


Figure 4.18: Reflux Classifier (courtesy of Ludowici)

The feed enters into the top inlet and into the upper section of the classifier below the lamella plates and directly above the fluidization chamber. Fluidization water is injected through several nozzles at the bottom of the unit at a flow rate that creates a fluidized bed of high density particles within the mixing chamber. This water is adjusted to ensure that the clean coarse particles in the feed are able to enter the lamella plates and leave with the overflow stream. A pair of density probes measures the density of the fluidized bed, allowing the operator to manipulate the operation of the unit to achieve the desired separation. The discard coal is rejected through the underflow valve.

Results from metallurgical coal tested by Ghosh et al (2011) showing the average epm and specific gravity cutpoints are presented in Table 4.10, where a significant shift in the cutpoint with decreasing size is observed, which explains the relative overall efficiency of 0.200. The unit was noted to be very efficient at sizes above 0.6mm, and better than typical spiral efficiencies for the 0.6x0.3mm size fraction.

Table 4.10: Summary of results for a metallurgical coal (Ghosh et al 2011)

Particle Size Fraction	d ₅₀	epm
+1.0mm	1.57	0.057
1.0x0.6mm	1.69	0.078
0.6x0.3mm	1.88	0.148
0.3x0.15mm	2.25	----
Overall	1.78	0.200

These results were found to be consistent with internal research work undertaken at Exxaro Metallurgy Research and Development by La Grange (2012), though the specific results cannot be reported due to their confidential nature. The Tshikondeni product could not be tested due to short life of mine once unit was properly commissioned during late 2012, the mine only having 2 to 3 years of reserves left at this point.

4.1.9 Enhanced Gravity Separation

Enhanced gravity separation can be used for the processing of fine coal, and also has been noted to be a viable alternative to froth flotation in the coarser sizes (>75µm) of the -0.6+0mm size fraction (Honaker et al, 1995). It was reported that the use of enhanced gravity separators may overcome the shortcoming of froth flotation and that the washability curves from density based separations have always been found more efficient at treating coals with a significant amount of middlings, this being due to these units providing more centrifugal force needed to treat this size fraction.

Commercially available units in this technology class are the Multi-Gravity Separator (MGS), the Kelsey JIG, the Knelson Concentrator, and the Falcon concentrator. Honaker et (1995) focussed his work on the Falcon concentrator, over the size range -0.6mm+10µm size range, which was able to achieve high rejections of both ash and

sulphur, though the latter would not be achievable for Tshikondeni due to the sulphur being predominantly bound to the organic matrix.

The Falcon Concentrator was trialled at pilot scale in South Africa in the mid to late 1990s, though was never installed at a colliery, due to the unit being represented by the major suppliers of coal spirals and thus were seen to be competing against themselves (Alston, 2009).

The unit is shown in Figure 4.19

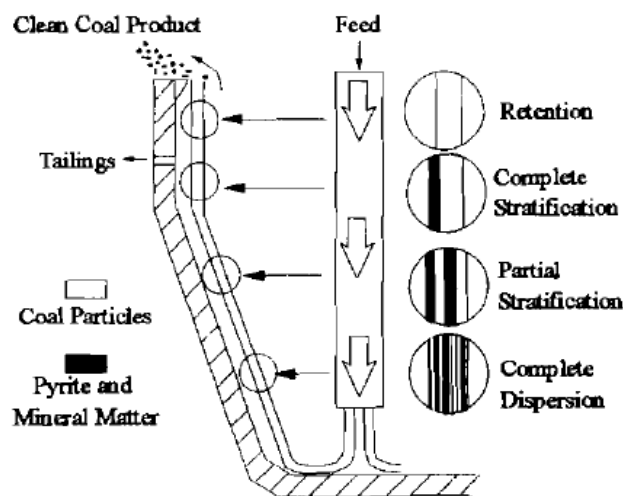


Figure 4.19: Falcon Concentrator (Honaker et al, 1995)

The concentrator is able to produce up to 300g of centrifugal force which causes deposition and stratification of the fine particles within the inside of a smooth centrifugal bowl. As can be seen in Figure 4.19 the feed enters the unit at the bottom of the bowl onto a spinning rotor. An impellor projects the feed against the rotor wall. The bottom of the rotor is known as the migration zone, and is inclined at a slight angle so that the enhanced gravity field generated by the rotor can be resolved into two force components. The strong component normal to the wall provides for the strong g-forces that create hindered settling and stratification of the feed articles. The second, weak, driving component parallel to the inclined rotor pushes the stratified solids up the wall to the top of the bowl. Light particles are ejected over the lip of the bowl, whilst the heavier and coarser particles are removed by a slot in the bowl wall.

Honaker and Wang (1998) describe work undertaken on Pittsburgh No. 8 seam coal in a Falcon C40 unit that could operate at approximately 100 tonnes per hour, which would be more than adequate to install at Tshikondeni's plant. The unit feed was nominally -0.6mm in size, but contained a high proportion of ultrafine -37 μ m material at 42% by mass. The -0.6+0.3mm fraction accounted for approximately 26% of the feed. The feed contained 33.0% ash and 3.40% total sulphur. The results achieved from a series of tests indicated that approximately 75% of the ash bearing and 60% sulphur were removed, whilst recovering 80% to the product at an organic efficiency of the order of 96%.

When the unit was compared to froth flotation, the +75 μ m material gave better process efficiencies with the Falcon, whilst the -75 μ m performed better using froth flotation. Similar to Tshikondeni, the processing of the entire sample (albeit finer than Tshikondeni) of -0.6mm, by the use of froth flotation resulted in a much inferior performance. Though not considered in Chapter 9, whereby options to reconfigure the plant were evaluated, it is evident that splitting out the coarser material to be processed using a gravity process and the finer fractions by froth flotation would be likely to give an overall improvement in plant performance.

4.1.10 *Three Product Cyclone*

The three product cyclone is a recent development in coal preparation in South Africa with two units in operation, the first at Umlalazi Colliery and the second at Exxaro's New Clydesdale Colliery. De Korte (2012) noted that the 3 product cyclone was developed during the late 1960's in Russia and pre-dates the similar but more well-known Larcodems separator. The three product cyclone is, however, widely applied in Russia and China. Locally it would be a direct competitor to the Tri-Flo separator discussed in section 4.1.2 above.

The unit can either be pump or gravity fed as depicted in Figure 4.20 and Figure 4.21 respectively. Umlalazi's unit is illustrated Figure 4.22, which clearly shows the primary and secondary stages. A wide variety of diameters, and hence capacities are available.

The primary cylindrical section produces a higher grade product, the rejects of which are then reprocessed in the secondary section (which looks like a conventional cyclone) to produce a lower grade (“middlings”) product. In Russia and China it is typically used to produce a coking coal and a middlings coal, whilst in South Africa it would have applications to:

- simultaneously produce a primary export product and a secondary product suitable for Eskom’s power generation needs from the Witbank coalfields,
- metallurgical coal and thermal coal from the Waterberg and Soutpansberg coalfields
- the unit also has the potential to destone a raw feed at a high operating density (~ 2.00) using a circulating medium in the order of 1.50.

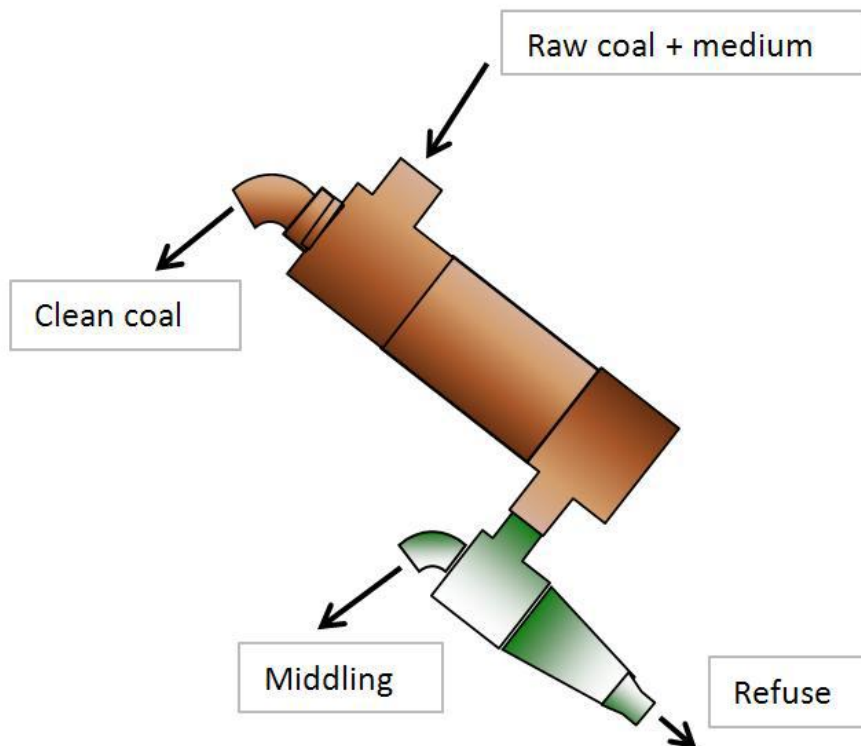


Figure 4.20: Pump fed 3 product cyclone (De Korte, 2012)

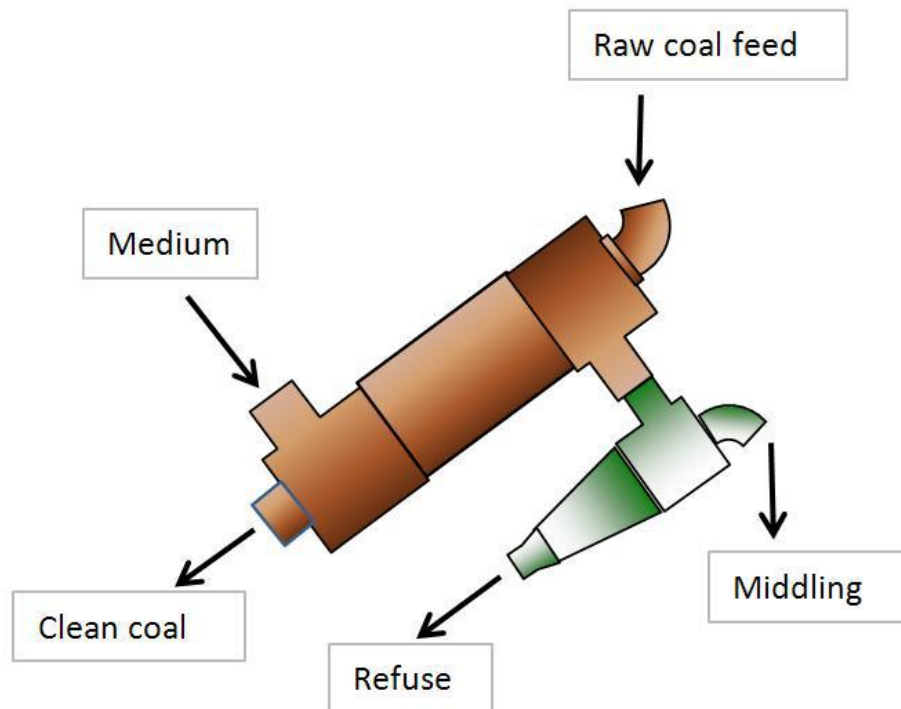


Figure 4.21: Gravity fed 3 product cyclone (De Korte, 2012)



Figure 4.22: Pump fed 3 product cyclone installed at Umlalazi Colliery (De Korte, 2012)

Performance data reported by De Korte (2012) on Russian installations for 350/250mm units are presented in Table 4.11.

Table 4.11: Summary of results for various Russian coal mines (De Korte, 2012)

Mine	Feed RD	Feed Size (mm)	d50/epm 1st Stage	d50/epm 2nd Stage
Karanganda Basin	1.36	13 x 0.5	1.430/0.020	1.960/0.032
YaKutsk Basin	1.39	14 x 0.5	1.430/0.030	1.930/0.050
Karanganda Basin	1.40	15 x 0.5	1.440/0.025	1.930/0.040
Donets Basin	1.55	6 x 0.5	1.650/0.050	2.100/0.060
Leningrad	1.35	6 x 0.5	1.410/0.040	1.820/0.055

The initial results from tests performed by De Korte at the Umlalazi operation are presented in Table 4.12 and Table 4.13 for the primary and secondary stages respectively on the whole unit stage and three size fractions. Note that the near density material for the secondary stage could not be calculated due to the high separation densities achieved.

Table 4.12: Summary of results for Umlalazi primary stage (De Korte, 2012)

Parameter	Plus 20 mm	20 x10 mm	10 x 1 mm	Combined 50 x1 mm
Feed Ash (%)	21.1	19.0	18.2	19.6
Product Ash (%)	11.9	11.3	11.0	11.4
Discard Ash (%)	40.5	36.0	34	37.1
Product Yield (%)	67.76	68.65	68.7	68.32
D50 Cut Point RD	1.532	1.527	1.527	1.530
Ep	0.0103	0.0118	0.0201	0.0144
Organic Efficiency	99.2	98.1	96	97.7
Sinks in Floats (%)	2.04	2.1	2.7	2.19
Floats in Sinks (%)	2.21	3.93	5.38	3.90
Total Misplaced (%)	4.24	6.04	8.08	6.09
Near Density Material (%)	33.7	38.1	36.3	35.2

Table 4.13: Summary of results for Umlalazi secondary stage (De Korte, 2012)

Parameter	Plus 20 mm	20 x10 mm	10 x 1 mm	Combined 50 x1 mm
Feed Ash (%)	40.5	36.0	34.0	37.1
Product Ash (%)	31.9	28.7	27.5	29.5
Discard Ash (%)	65.1	59.8	57.5	61.4
Product Yield (%)	74.09	76.89	78.33	76.11
D50 Cut Point RD	1.983	1.928	1.911	1.970
Ep	0.0096	0.0279	0.0313	0.0264
Organic Efficiency	99.7	98.7	96.9	98.3
Sinks in Floats (%)	2.21	0.64	1.49	1.53
Floats in Sinks (%)	1.02	1.88	3.37	2.67
Total Misplaced (%)	3.23	2.52	4.86	4.20
Near Density Material (%)	-	-	-	-

It can be seen that overall good efficiencies were achieved and that the unit has huge potential within the South African coal preparation industry to maximize the utilization of the dwindling coal resources.

One current problem noted with the three product cyclone is the lack of density control of the secondary section, and further development in the area is required to allow for some adjustment of the product quality from the secondary stage.

From the discussion above, the obvious question would be as to whether the Installation of this unit at Tshikondeni would have the potential for the secondary stage to produce either a middlings product for thermal use, or allow for capture of coking coal lost to the discard dump for re-processing by milling and flotation?

A simulation for both shafts was therefore performed to explore this possibility using the washability data presented later in this work under Chapter 8 on run of mine samples sourced from both Mutale and Vhukati Shafts. The simulation was carried out using the Whiten equation (Equation 1) shown below:

$$P = \frac{1}{1 + \exp\{1.0986(d_{50} - d)/Ep\}} \dots\dots\dots \text{Equation 4.1}$$

Where P is the partition coefficient, d_{50} is the separation size and d is the specific size and Ep is the ecart probable moyen.

The primary separation targeted a primary coking coal ash content of 14.0% as per the standard Tshikondeni specification. The simulation was performed using the Whiten Equation with a conservative E_p value of 0.025 being used and the d_{50} manipulated to give the 14.0% ash product. The respective separation densities (d_{50}) were determined to be 1.54 and 1.59 for Mutale Shaft and Vhukati Shaft respectively. The results for the primary separation are detailed in Table 4.14 and Table 4.15 for Mutale and Vhukati shafts respectively. Note that the yields and ashes shown in both tables are fractional values, whilst the total line shows the cumulative figures.

Table 4.14: Primary separation Simulation for Mutale Shaft for the three product cyclone

RD Range		Mean RD	Input Data		Simulated Partition Number	Cyclone Product		Cyclone Discard	
Sinks	Floats		Yield	Ash		Yield (%)	Ash (%)	Yield (%)	Ash (%)
-	1.30	1.15	11.3	2.1	1.000	11.3	2.1	0.0	2.1
1.30	1.35	1.33	13.6	8.9	1.000	13.6	8.9	0.0	8.9
1.35	1.40	1.38	14.4	13.9	0.999	14.4	13.9	0.0	13.9
1.40	1.45	1.43	11.7	19.4	0.993	11.6	19.4	0.1	19.4
1.45	1.50	1.48	8.2	23.4	0.943	7.8	23.4	0.5	23.4
1.50	1.55	1.53	5.7	29.0	0.649	3.7	29.0	2.0	29.0
1.55	1.60	1.58	3.4	32.4	0.171	0.6	32.4	2.8	32.4
1.60	1.65	1.63	2.8	35.6	0.022	0.1	35.6	2.8	35.6
1.65	1.70	1.68	2.3	38.8	0.003	0.0	38.8	2.3	38.8
1.70	1.75	1.73	2.3	43.7	0.000	0.0	43.7	2.3	43.7
1.75	1.80	1.78	2.1	46.6	0.000	0.0	46.6	2.1	46.6
1.80		1.83	22.1	72.0	0.000	0.0	72.0	22.1	72.0
Total			100.0	30.2		62.9	14.0	37.0	57.9

Table 4.15: Primary separation Simulation for Vhukati Shaft for the three product cyclone

RD Range		Mean RD	Input Data		Simulated Partition Number	Cyclone Product		Cyclone Discard	
Sinks	Floats		Yield	Ash		Yield (%)	Ash (%)	Yield (%)	Ash (%)
-	1.30	1.15	12.7	1.9	1.000	12.7	1.9	0.0	1.9
1.30	1.35	1.33	14.7	7.9	1.000	14.7	7.9	0.0	7.9
1.35	1.40	1.38	14.6	12.9	1.000	14.6	12.9	0.0	12.9
1.40	1.45	1.43	12.7	18.9	0.999	12.6	18.9	0.0	18.9
1.45	1.50	1.48	8.4	23.0	0.994	8.3	23.0	0.1	23.0
1.50	1.55	1.53	5.3	27.9	0.946	5.1	27.9	0.3	27.9
1.55	1.60	1.58	3.3	31.7	0.659	2.2	31.7	1.1	31.7
1.60	1.65	1.63	2.4	33.8	0.177	0.4	33.8	2.0	33.8
1.65	1.70	1.68	2.0	39.8	0.023	0.0	39.8	1.9	39.8
1.70	1.75	1.73	1.9	41.7	0.003	0.0	41.7	1.9	41.7
1.75	1.80	1.78	1.8	45.0	0.000	0.0	45.0	1.8	45.0
1.80		1.83	20.1	69.7	0.000	0.0	69.7	20.1	69.7
Total			100.0	27.4		70.7	14.0	29.3	59.9

Primary yields of approximately 63% and 71% for Mutale Shaft and Vhukati Shaft respectively were predicted and are slightly higher than would normally be expected from these sources, though it was assumed that all size fractions were processed through the unit i.e. wash to zero, which is also discussed in detail in Chapter 8.

The simulated primary stage discards would be expected to be in the order of 58% ash and 60% ash for Mutale Shaft and Vhukati Shaft respectively. A more detailed examination of the discard for both shafts shows that mass content in the relative density ranges between 1.40 to 1.80 are very low, and therefore virtually no middlings product would be produced in the secondary stage. It is therefore evident that the run of mine material from both Mutale Shaft and Vhukati Shaft would not be amenable to processing by a three product cyclone.

Therefore, it could be concluded that the use of a three product cyclone at Tshikondeni would not be appropriate.

4.2 Destoning Experimental Procedures

4.2.1 Dump Borehole Samples

As detailed in 3.2.2 above, for each borehole, the 1.5m long incremental core lengths were combined to form composite samples of 4.5m long. The composite samples were then air dried, weighed, mixed well and split so that three quarters of the sample was available for testing and the remaining quarter stored as a reference sample for any future work.

The analytical sample was then processed according to the procedure illustrated in Figure 4.23. The same procedure was followed for both the old and new dumps.

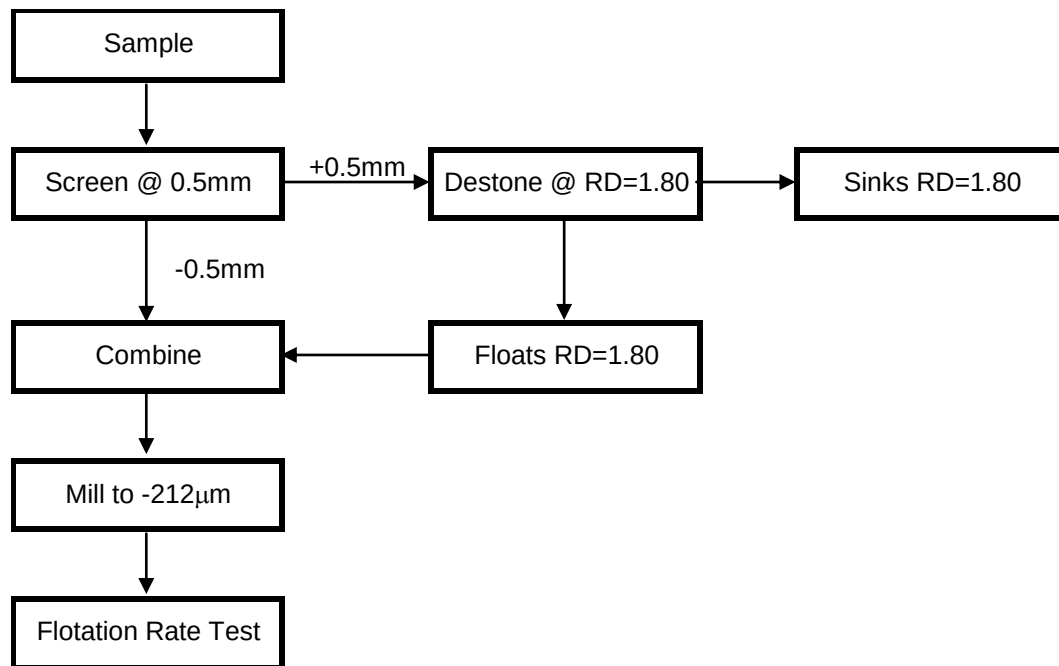


Figure 4.23: Borehole sample test procedure

Each 4.5m composite sample was screened at 0.5mm, the +0.5mm fraction was then destoned by subjecting it to a single density float and sinks step at a relative density of 1.80. The floats fraction was then combined with the -0.5mm fraction, milled to -212µm and subjected to a flotation rate test, which will be discussed in detail in Chapter 6. At each stage, a representative analytical sample was extracted, prepared and the ash content determined according to the relevant ISO standards.

4.2.2 Current Arising Discard – Monthly Composite Samples

The current arising discard was examined by means of composite samples made up from the daily production samples for the months of April 2008, May 2008, June 2008 and September 2008.

The Tshikondeni plant has an automatic swing hammer sampler on the discard belt which generates the hourly discard sample used for process control purposes. The separate hourly samples generated are combined to form a daily composite, these are then used to form a weekly sample, which in turn is used to produce the final

monthly discard composite. The samples from the above mentioned months were subjected to a more detailed examination than that which normally occurs.

The April 2008 and May 2008 samples were treated in a manner similar to that of the borehole samples described in the previous section. It was noticed, though, that significant amounts of +13mm material were present in the samples. This would be expected in an open loop crushing circuit such as that used at Tshikondeni: i.e. the more friable coal would be expected to crush preferentially over any gangue minerals present in the run of mine material. The test process with the crushing of the +13mm material is illustrated in Figure 4.24.

For the June 2008 and August 2008 samples, a comprehensive screening exercise was included in the test procedure. Each sample was screened at 20mm, 18mm, 12.5mm, 8mm, 3mm, 1.4mm, 0.6mm, 212 μ m, 125 μ m and 75 μ m. Thereafter the +20mm, -20+18mm and -18+12.5mm fractions were recombined and crushed to -13mm and the test procedure adopted for the April 2008 and May 2008 samples as illustrated in Figure 4.24 carried out.

For comparative purposes, the -1.4mm material was also subjected to a flotation rate test to simulate what could be recovered under normal plant conditions. For the June and August samples, the arising -212 μ m was also subjected to a flotation rate test. This is discussed in greater detail later in the work detailed in section 6.3.2.

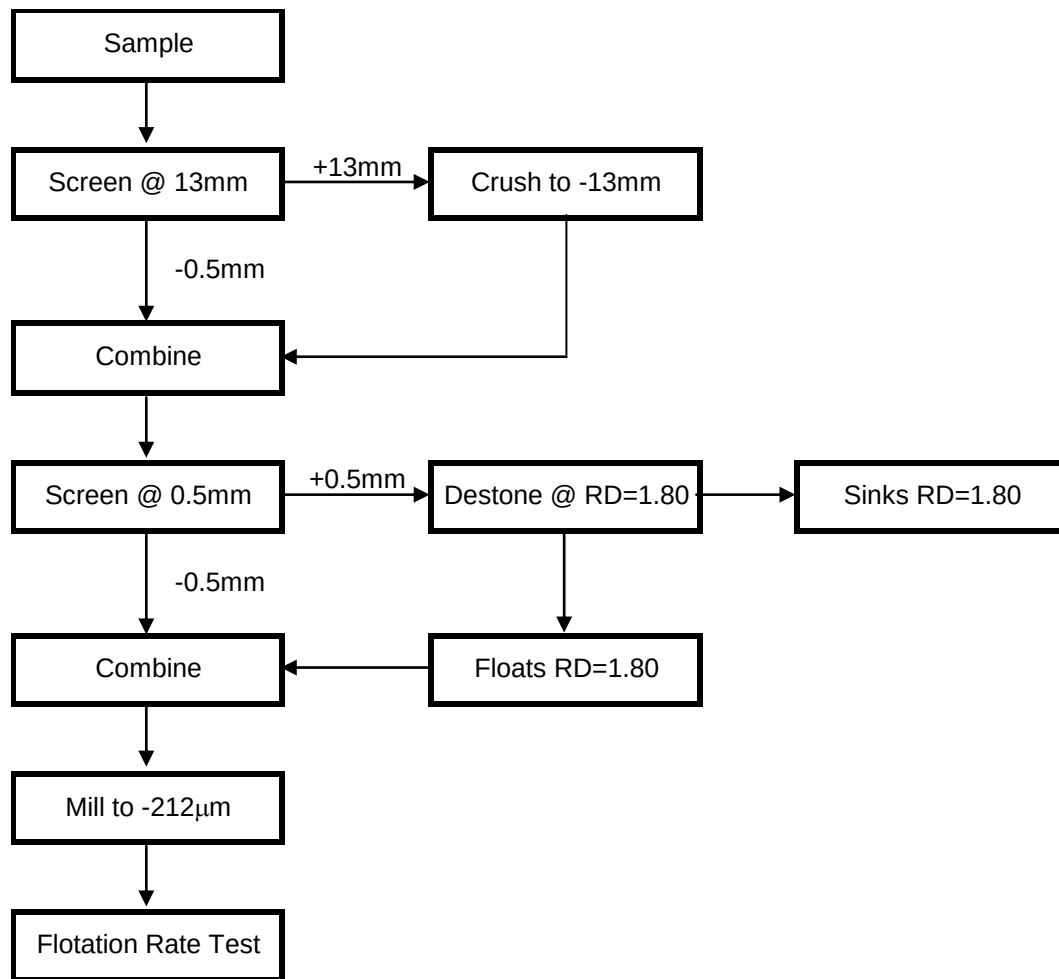


Figure 4.24: Arising discard test procedure

4.2.3 Current Arising Discard – 1 Tonne Bulk Samples

A series of five bulk samples of approximately 1 tonne each were collected in October 2008. The samples were made up of the material collected by the automatic hammer sampler used to take the hourly plant quality control samples. The samples were taken according to the following concept:

- Two separate samples from discard produced when the plant was processing ROM from Mutale shaft. Mutale shaft typically has the lowest yield in the plant and can be problematic in terms of processing, therefore this shaft has the largest potential for the recovery of values lost to the discard stream and

hence the collection of a pair of samples. The samples were identified as “Mutale Shaft” and “Mutale Plant Waste”.

- Two separate samples from discard produced when the plant was processing ROM from Vhukati shaft. Vhukati can be considered to be the “average” shaft i.e. the behaviour in the plant is typically better than Mutale, but not as good as Goni. Hence two samples were also collected from this shaft. The samples were identified as “Vhukati Shaft” and “Vhukati Plant Waste”.
- One sample from discard produced when the plant was processing ROM from Goni shaft. Goni shaft typically produces the highest yields and generally processes the best of the three shafts, and would therefore be the discard that would probably be the least amenable to further processing. The sample was labelled “Goni Plant Waste”.

The samples were subjected to a series of tests, with the test procedures illustrated in Figure 4.25 to Figure 4.27. Each sample was split to give three sub-samples, one to be used for the milling testwork to be carried out at Mintek (discussed in detail in Chapter 5), one for detailed characterization in terms of size distributions and washability behaviour of the various size classes, and a reference sample.

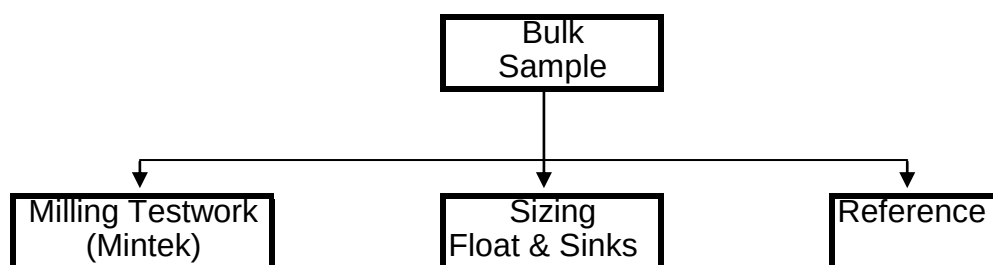


Figure 4.25: High level outline of work undertaken

The milling test sample was screened at 0.5mm, the +0.5mm fraction was destoned by a process of float and sinks at a single relative density fraction of 1.75. This slight change in relative density was intended to give an overall improvement of the quality of the destoned fraction and should produce a better flotation product compared to previous tests performed to this point. The material that floated at RD=1.75 was

combined with the -0.5mm screen fraction and sent to Mintek for the milling tests (discussed in detail in Chapter 5) as illustrated in Figure 4.26.

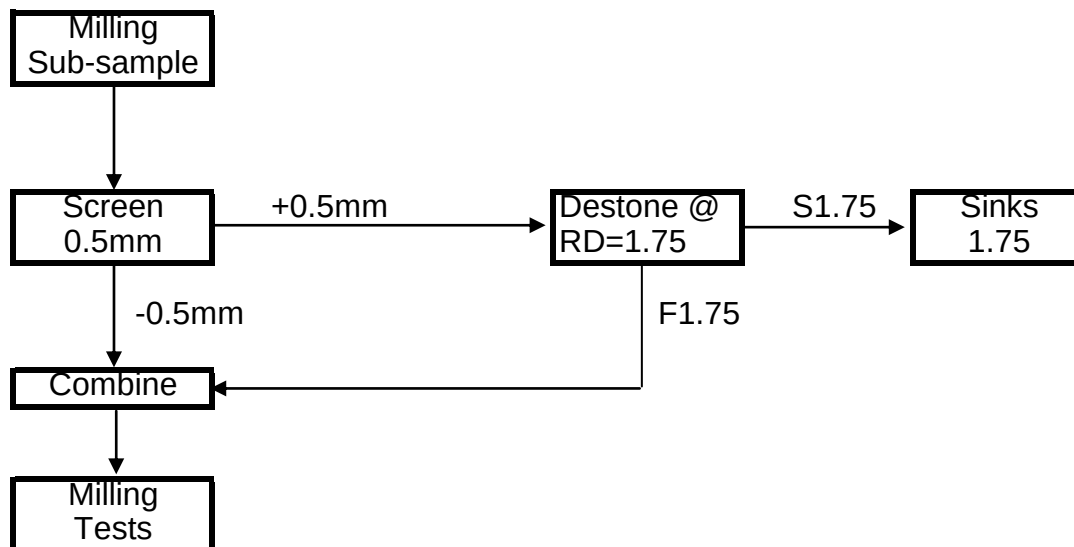


Figure 4.26: Sample preparation prior to milling testwork

A detailed characterization exercise was also undertaken to determine the washability characteristics of the sample over a range of size classes, as depicted in Figure 4.27. The sample was initially screened at 16mm, 8mm, 4mm, 2mm, 1mm and 0.5mm and the size distribution determined. Thereafter, each size class, except for the -0.5mm material, was subjected to a float and sinks exercise starting at a relative density of 1.30 and increasing in 0.05 increments up to a final relative density of 1.90. Each fraction, including the sinks 1.90 was analyzed for inherent moisture and ash content to allow for the determination of the ash content on a dry basis. The -0.5mm fraction was analyzed for inherent moisture and ash contents only.

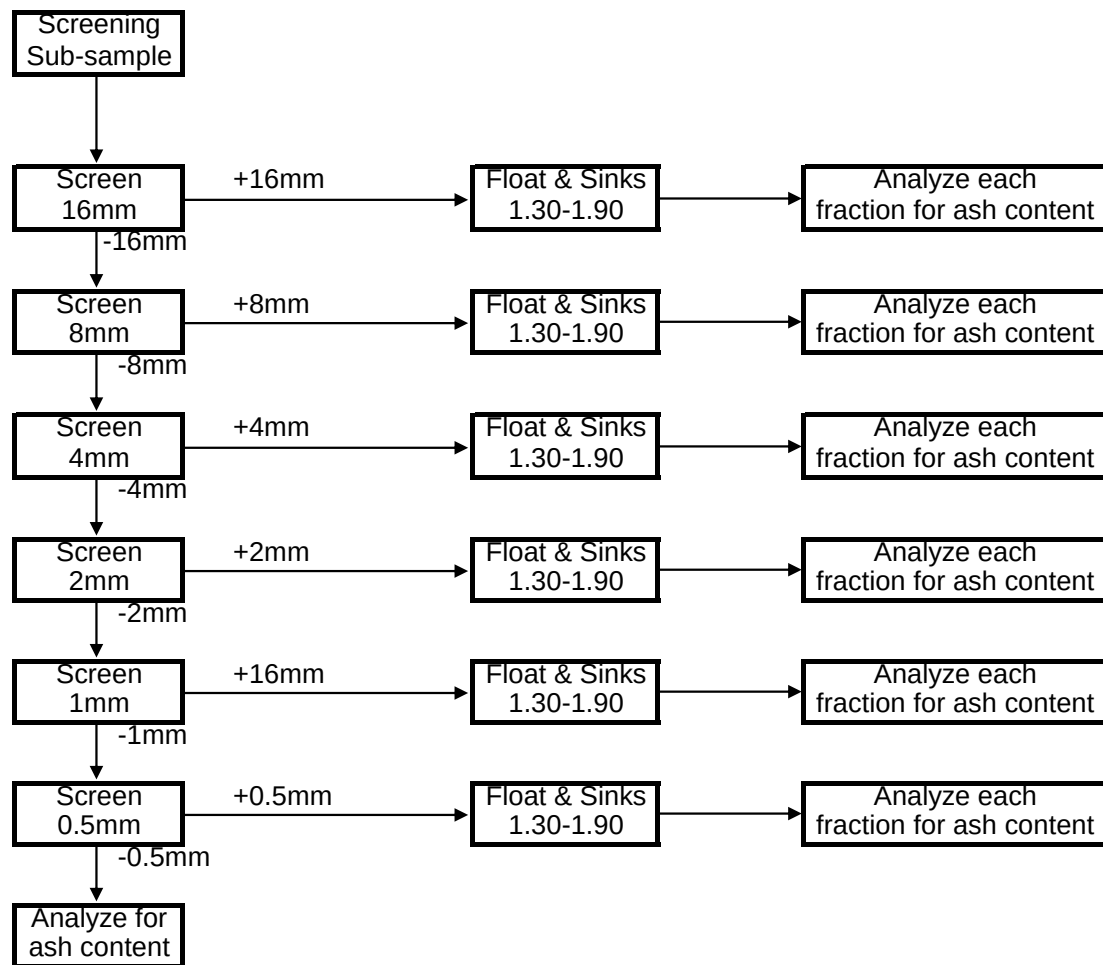


Figure 4.27: Detailed characterization in terms of sizing and washability

4.2.4 Old Discard Dump- 20 Tonne Bulk Sample

The final bulk sample examined was approximately 20 tonnes in mass and was taken from an easily accessible section of the dump. This section of the dump had been well compacted, but not yet rehabilitated. It was assumed that any oxidation/weathering effects would be apparent in the material close to the surface, and therefore the first metre or so of the material was dozed away. Thereafter, the sample was collected using a front end loader and the sample extracted to a depth of around 5m in an attempt to try and obtain a reasonable representation of the material in the dump. The material was then bagged in 1m³ bulk bags and sent to

the Exxaro Research and Development department for processing. Figure 4.28 and Figure 4.29 show the hole left behind in the dump after collection of the sample.



Figure 4.28: View of the dump after sample collection



Figure 4.29: Alternative view of the dump after sample collection

Approximately 5 tonnes of material was split out from the 20 tonnes of sample and processed using The FGX1 Separator, capable of handling 7-10 tonnes per hour of feed. The unit was being tested on several coal sources at another Exxaro site and a window of opportunity to test the Tshikondeni material on this unit arose at the end of the campaign. Only one test could be performed due to project time and resource constraints. The unit was operated with the three fan blowers set at 70%, 100% and 50% respectively. The table inclination from the back to the front was 8 degrees, along the length of the table the inclination was set to 0 (zero) degrees, i.e. no inclination, and the discharge lip height was set to maximum. From previous work performed on other coals (de Korte, 2009), the middlings were directed to the product sample.

Samples of the feed (from the vibrating feeder), product and discard (both from the respective discharge chutes) were taken and the respective ash contents determined. These were determined to be 55.1%, 50.3% and 71.2% and therefore, very little upgrade was achieved; hence a limited washability test was performed on the samples at relative densities of 1.40, 1.50, 1.60, 1.70, 1.75, 1.80 and Sinks 1.80.

The remainder of the sample was used for a more extensive flotation study to further explore the potential to make a saleable product by this technique. This work is discussed in Chapter 6.

4.3 Destoning Results and Discussion

4.3.1 Old Dump Borehole Samples

The detailed results are included in Appendix 2, with a summary of the effect of screening at 0.5mm and destoning of the +0.5mm at a relative density of 1.80 being presented in Table 4.16.

Table 4.16: Summary of old dump screening and destoning results

	Average		Standard Deviation		Maximum		Minimum	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
Insitu Discard	100.0	50.3	-	2.8	100.0	61.6	100.0	40.4
+0.5mm Material	76.4	53.4	5.0	3.0	96.3	65.0	57.8	39.8
+0.5mm Floats RD1.80	34.3	29.8	4.4	1.9	51.5	38.6	18.5	27.6
+0.5mm Sinks RD1.80	41.9	72.9	5.5	1.1	61.5	78.0	19.0	70.2
-0.5mm Material	23.6	38.6	5.0	5.1	42.2	48.1	3.7	23.1
Flotation Feed	57.9	34.1	5.6	2.1	81.0	43.5	38.5	29.1

The benefit of screening at 0.5mm is evident in that the in situ ash is reduced from an average value of 50.3% to 38.6% in the -0.5mm fraction and accounted for some 23.6% of the in situ mass. The average +0.5mm ash was noted to be 53.4%, which could be upgraded to 29.8% after separating at a relative density of 1.80. The floats product contained some 34.3% of the mass compared to 41.9% for the sinks fractions, which returned an ash content of 72.9%, suggesting that the destoning process had been effective.

By combining the destoned +0.5mm material with the -0.5mm material, a mill feed (flotation feed) of 57.9% in mass terms at an average ash content of 34.1% could be achieved.

4.3.2 New Dump Borehole Samples

The detailed results are included in Appendix 2, with a summary of the effect of screening at 0.5mm and destoning of the +0.5mm at a relative density of 1.80 being presented in Table 4.17.

Table 4.17: Summary of old dump screening and destoning results

	Average		Standard Deviation		Maximum		Minimum	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
Insitu Discard	100.0	55.8	-	1.7	100.0	58.0	100.0	52.4
+0.5mm Material	81.7	58.7	1.2	1.6	83.0	60.8	79.4	55.3
+0.5mm Floats RD1.80	28.0	30.4	2.5	0.9	33.0	31.5	25.0	28.8
+0.5mm Sinks RD1.80	53.7	73.4	2.9	0.8	57.8	74.7	48.0	72.2
-0.5mm Material	18.3	42.9	1.2	2.1	20.6	47.0	17.0	40.2
Flotation Feed	46.3	35.4	3.0	1.1	52.1	36.8	42.2	33.2

These samples were observed to contain less -0.5mm than the old dump (18% cf. 24%), destoning rejected some 54% of the in situ mass at an acceptable 73% ash. The destoned material represented approximately 28% of the in situ mass with an ash content of 30%, and when combined with the -0.5mm material generated an average flotation feed of 46% of the initial mass at an ash of 35%.

4.3.3 Current Arising Discard – Monthly Composite Samples

The detailed results for the current arising discard samples taken over the period April 2008 to August 2008 are included in Appendix 4, whilst Table 4.18 gives a summary of the results achieved.

Table 4.18: Summary of results

Stream	April 2008		May 2008		June 2008		August 2008	
	Mass (%)	Ash (%)	Mass (%)	Ash (%)	Mass (%)	Ash (%)	Mass (%)	Ash (%)
Feed	100.0	54.7	100.0	57.7	100.0	55.3	100.0	57.6
Floats 1.80	37.2	29.2	33.2	30.3	34.8	28.0	31.5	27.6
Sinks 1.80	57.7	71.6	65.4	71.9	60.2	71.6	62.9	73.1
-0.5mm	5.4	46.3	1.4	43.0	5.0	47.8	5.6	53.2
Flotation Feed	42.6	31.4	34.6	30.8	39.8	30.5	37.1	31.5

The feed ashes were in the order of 54-58% which after screening at 0.5mm and floating at a relative density of 1.80, upgraded to 28-30% ash. The destoning stage accounted for a mass reduction of the 58-65% at ashes of approximately 72-73%. The -0.5mm accounted for less than 6% of the mass, with ashes ranging between 43-53%.

The flotation feed, comprised of the -0.5mm fraction and floats of the 1.80 material accounted for between 35-43% of the original mass with an ash content of approximately 31%. The flotation performance is discussed in Chapter 6 in detail.

The full size distributions for the June 2008 and August 2008 samples are illustrated in Figure 4.30. The August 2008 sample was noted to be generally finer than the June 2008 sample. It was also noted that a significant proportion of material was found to be larger than the plant's nominal top size of 13mm, with amount of approximately 25% and 32% being determined for the June 2008 and August 2008 samples respectively. This would be expected since the discard from Tshikondeni tends to be flat and rectangular in shape and would therefore tend to be retained on a laboratory sieve, as illustrated Figure 4.31.

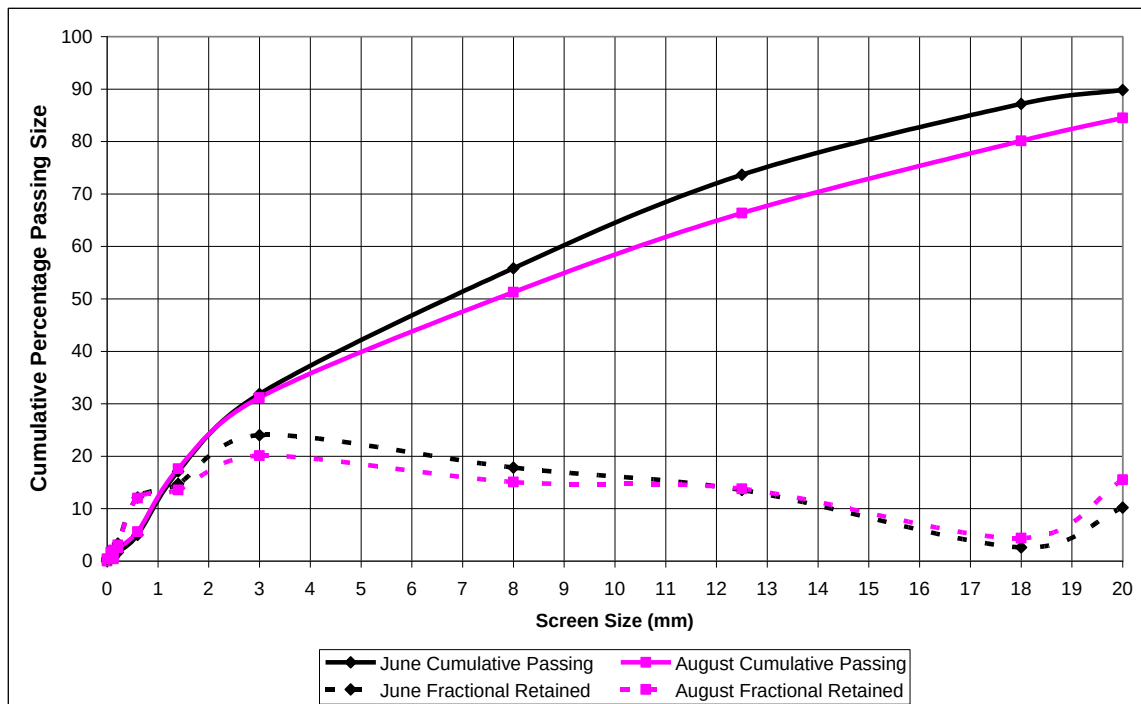


Figure 4.30: Size distributions of the June 2008 and August 2008 monthly composites



Figure 4.31: Cyclone discard showing the flat, slab like nature of the material

4.3.4 Current Arising Discard – 1 Tonne Bulk Samples

This section should be read in conjunction with the following chapter which examined the determination of the milling characteristics of the discard after being destoned.

The detailed results from the screening and washability by size exercise are included in Appendix 4. The head ash content, measured on a dry basis, for the five samples is summarized in Table 4.19.

Table 4.19: Bulk sample head ashes

Sample	Ash Content (% dry basis)
Mutale Shaft	56.6
Mutale Plant Waste	60.9
Vhukati Shaft	60.7
Vhukati Plant Waste	57.5
Goni Plant Waste	59.4
Average	59.0

The plant discard was noted to have an average ash content of 59%, with a variation of between 56.6% (Mutale Shaft) and 60.9% (Mutale Plant Waste), and were found to be consistent with historical plant data.

The comparative results from the screening exercise are presented in Figure 4.32, an average size distribution was also calculated. The plant treats a nominal top size of 13mm and this has been used as a reference point on the graph.

The Vhukati Shaft sample was observed to “stand out” from the other four samples as the coarsest samples in the set. Approximately 54% of the sample was determined to be coarser than 13mm. Plant discard samples tend to be flat and plate like rather than the cubic shape observed in plant product, and would therefore be expected to pass through the plant’s open circuit crusher. (Refer to Figure 4.31 in the previous section)

The other four samples and the average, were observed to be similar in size distribution with 35-40% of the material being larger than 13mm. The Goni sample being the finest with 35% retained at 13mm. The Mutale Plant Waste sample at 40%

retained was observed to be similar to the mathematical average of 39% retained at 13mm.

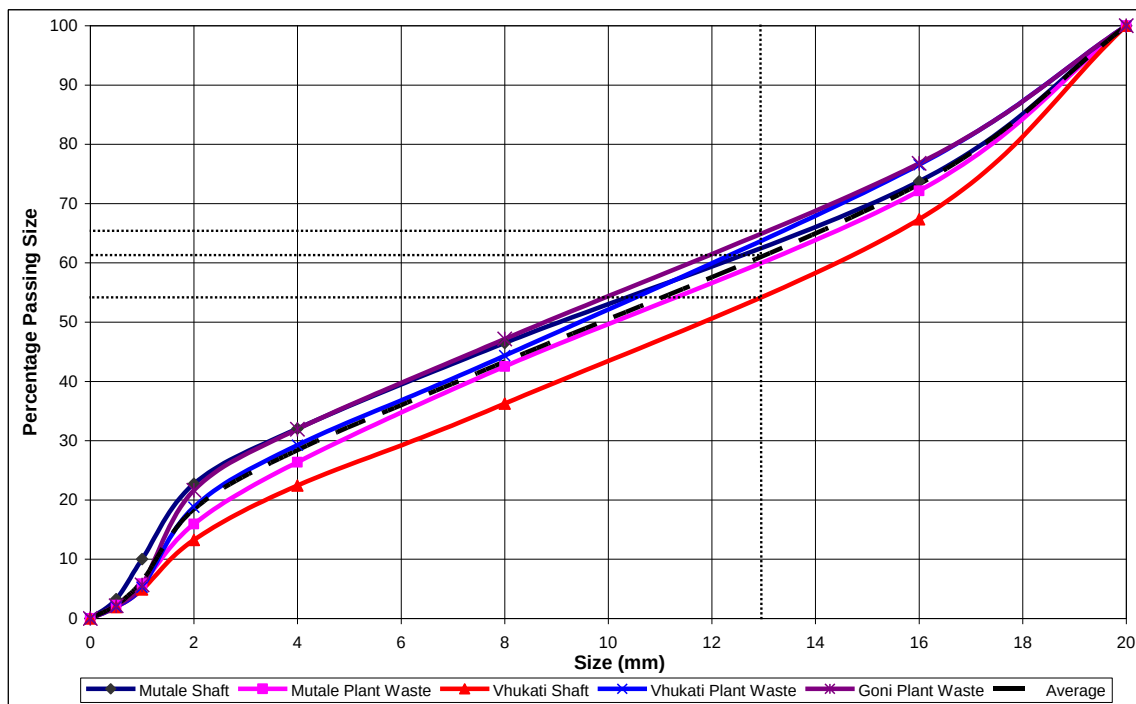


Figure 4.32: Bulk sample screening results

The comparison of the five samples in terms of destoning the whole +0.5mm material at a relative density of 1.75 is presented in Table 4.20, along with the calculated average. The average indicated that approximately 38% of the material in the discard could be recovered at a dry ash content of approximately 25.8%

Table 4.20: Bulk sample destoning results at RD = 1.75

Sample	Yield (%)	Ash Content (% dry basis)
Mutale Shaft	39.3	27.2
Mutale Plant Waste	38.3	24.6
Vhukati Shaft	37.3	26.7
Vhukati Plant Waste	38.2	26.6
Goni Plant Waste	35.8	23.7
Average	37.8	25.8

The effect of destoning the samples at a relative density of 1.90 is summarized in Table 4.21, which indicated that the yield would increase to an average of approximately 49% at an ash content of approximately 32%.

Table 4.21: Bulk sample destoning results at RD = 1.90

Sample	Yield (%)	Ash Content (% dry basis)
Mutale Shaft	49.9	32.5
Mutale Plant Waste	48.1	30.1
Vhukati Shaft	49.1	32.8
Vhukati Plant Waste	50.5	32.9
Goni Plant Waste	46.4	30.5
Average	48.8	31.8

Both Table 4.20 and Table 4.21 show that a significant portion of the discard can be rejected as a final discard prior to re-treatment, and that a substantial upgrade of the material can be achieved by destoning. Moreover, should further upgrade of the material not be achievable the potential to make the destoned product available to the open market for supply to Eskom would be worth considering as an option.

Given that the destoning process would require either dense medium separation or a jigging process, an alternative method of effecting the removal of the higher ash material would be to consider screening the discard prior to destoning, which would be a cheaper option in terms of both capital and operating costs. A summary of the mass yields and ash values by size class is presented in Table 4.22.

Table 4.22: Yield and ash by size fraction

Size Fraction (mm)		Mean	Mutale Shaft		Mutale Plant Waste		Vhukati Shaft		Vhukati Plant Waste		Goni Plant Waste	
Upper	Lower	Size (mm)	Yield	Ash	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
20.0	16.00	17.89	26.3	69.8	27.8	75.7	32.7	71.5	23.5	69.9	23.2	74.1
16.0	8.00	11.31	27.3	60.3	29.6	62.9	31.1	61.8	32.2	61.8	29.6	66.8
8.0	4.00	5.66	14.4	56.1	16.2	58.5	13.8	58.5	15.1	58.2	15.2	63.5
4.0	2.00	2.83	9.3	53.6	10.4	54.7	9.2	54.8	10.5	52.8	10.3	55.3
2.0	1.00	1.41	12.7	39.8	10.1	39.1	8.4	38.9	13.5	36.7	16.0	32.4
1.0	0.50	0.71	6.7	40.5	3.6	39.1	3.0	40.0	3.3	41.6	3.5	36.6
0.5	0.00	0.01	3.3	28.2	2.2	29.3	1.9	27.6	2.0	26.5	2.1	27.0
Total			100.0	56.6	100.0	60.9	100.0	60.7	100.0	57.5	100.0	59.4

From Table 4.22 it is evident, as would be expected, that the higher ash material lies within the coarser size fractions, particularly the +16mm fraction (70-76% ash) and to a lesser degree in the -16+8mm fraction (60-67% ash). Combined, these two fractions for each of the shafts account for approximately 60% of the mass of the sample, and a significant upgrade of the material could be achieved by removal of the +8mm by screening alone as presented in Table 4.23, it can be observed that the ash content of the various discard samples is reduced from between 57% to 61%

ash to between 47% to 50% ash. This is discussed further in Section 4.4 where the various flowsheet options are examined and screen separations are simulated for the various sources.

Table 4.23: Effect of removal of the +8mm material

Size Fraction (mm)		Mean	Mutale Shaft		Mutale Plant Waste		Vhukati Shaft		Vhukati Plant Waste		Goni Plant Waste	
Upper	Lower	Size (mm)	Yield	Ash	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
8.0	4.00	5.66	14.4	56.1	16.2	58.5	13.8	58.5	15.1	58.2	15.2	63.5
4.0	2.00	2.83	9.3	53.6	10.4	54.7	9.2	54.8	10.5	52.8	10.3	55.3
2.0	1.00	1.41	12.7	39.8	10.1	39.1	8.4	38.9	13.5	36.7	16.0	32.4
1.0	0.50	0.71	6.7	40.5	3.6	39.1	3.0	40.0	3.3	41.6	3.5	36.6
0.5	0.00	0.01	3.3	28.2	2.2	29.3	1.9	27.6	2.0	26.5	2.1	27.0
Total			46.5	46.9	42.5	49.8	36.3	49.9	44.3	47.7	47.1	47.5

Having considered the removal of a significant portion of material by screening, the next logical step was to examine what size range would be most amenable to dense medium separation. From the data in Appendix 5 it was noted that for the total washability results from each sample, that a product of the order of 14% to 15% ash could be obtained by washing at a relative density of 1.50, as presented in Table 4.24.

Table 4.24: Bulk sample washing results at RD = 1.50

Sample	Yield (%)	Ash Content (% dry basis)
Mutale Shaft	16.3	15.3
Mutale Plant Waste	19.3	14.4
Vhukati Shaft	16.8	14.6
Vhukati Plant Waste	17.2	14.5
Goni Plant Waste	21.1	13.6
Average	18.1	14.5

Obviously the effect of size needed to be considered and a comparison of the yield and ash obtained by washing each size at a relative density of 1.50 is summarized in Table 4.25. From this table it is noted that higher fractional yields and better ashes are obtained as the size decreases, and that little benefit from dense medium separation would be seen in the -20+16mm and -16+8mm size fractions. These results seem to concur with the screening results above, and that the potential to produce a saleable product by screening at 8mm followed by washing at a relative density in the order of 1.50 could be an option. However, the overall plant yield

would be low and may not be economically viable. This option will also be explored in more detail in Chapter 5.

Table 4.25: Bulk sample comparative washing results at RD = 1.50 by size fraction

Size Fraction (mm)		Mutale Shaft		Mutale Plant Waste		Vhukati Shaft		Vhukati Plant Waste		Goni Plant Waste	
Upper	Lower	Yield	Ash	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
20.0	16.00	1.1	25.5	0.9	23.8	0.2	23.8	0.2	25.5	0.1	18.1
16.0	8.00	3.8	24.1	4.6	22.5	1.9	23.3	0.9	21.8	0.7	17.0
8.0	4.00	5.6	22.6	6.8	22.0	3.0	21.8	2.4	20.7	2.8	16.6
4.0	2.00	9.2	18.0	14.1	16.4	11.9	15.7	14.4	14.5	17.2	13.4
2.0	1.00	41.5	13.2	44.6	12.7	42.4	12.9	46.8	13.3	56.9	12.5
1.0	0.50	36.4	14.7	44.7	13.4	41.2	14.9	38.7	15.3	48.8	14.8
Total (based on feed)		16.3	15.3	19.3	14.4	16.8	14.6	17.2	14.5	21.1	13.6

The -8+4mm, -4+2mm, -2+1mm and -1+0.5mm washability by size data were combined together in mass proportions and the resulting cumulative washability results for the five samples are summarized in Table 4.26.

Table 4.26: Summary of washability data on the combined -8+0.5mm material

RD	Mutale Shaft		Mutale Plant Waste		Vhukati Shaft		Vhukati Plant Waste		Goni Plant Waste	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
Float 1.30	4.9	5.8	5.3	5.6	4.8	5.6	4.3	4.7	10.1	5.9
Float 1.35	7.6	7.1	8.4	6.9	7.0	6.8	9.4	7.4	15.9	7.9
Float 1.40	10.9	9.0	11.1	8.5	10.4	9.2	12.4	8.8	20.8	9.7
Float 1.45	16.3	12.0	15.3	11.1	13.8	11.5	16.9	11.1	25.0	11.4
Float 1.50	21.7	14.8	21.3	14.7	18.2	14.4	22.2	14.0	28.5	13.0
Float 1.55	28.5	18.2	26.6	17.3	23.5	17.7	27.2	16.7	31.6	14.6
Float 1.60	34.2	20.5	32.2	19.9	26.8	19.5	32.2	19.2	35.1	16.5
Float 1.65	38.0	22.1	36.9	21.9	32.4	22.5	36.6	21.3	38.3	18.2
Float 1.70	43.2	24.4	41.5	23.9	37.4	25.0	41.8	23.6	41.5	20.0
Float 1.75	47.2	26.0	43.9	25.0	41.8	26.9	45.9	25.4	45.0	22.0
Float 1.80	52.1	28.2	48.1	26.8	47.0	29.3	50.4	27.4	47.9	23.6
Float 1.85	55.9	30.0	52.4	29.0	50.4	30.8	55.3	29.7	52.4	26.1
Float 1.90	58.5	31.1	54.5	30.0	54.6	32.7	58.4	31.1	55.9	28.1
Sink 1.90	41.5	72.8	45.5	76.3	45.4	73.4	41.6	73.6	44.1	74.7
Total	100.0	48.4	100.0	51.1	100.0	51.2	100.0	48.8	100.0	48.6

It could also be the case that having removed a significant portion of material by screening, that the next logical step was to examine what could potentially be achieved from the -8mm material in terms of dense medium separation, i.e. whether to produce a 14% product directly at a low yield or consider the destoning option prior to milling and froth flotation. The former concept is explored further in Section 4.4 of this chapter.

4.3.5 Old Discard Dump- 20 Tonne Bulk Sample

The 5 tonnes of material tested on the FGX Separator had a feed ash of 55.1% and was able to generate a product ash of 50.3% (product plus middlings) at a yield of 77.0% with a discard ash content of 71.2%. Based on these results it was evident that the required level of upgrade of the discard dump material would not be achieved, and is most likely attributable to the size of the material being nominally -13+1.4mm. Therefore only a limited washability exercise was undertaken to obtain some basic characterization data. It is doubtful that any significant improvement could be achieved by varying the operational parameters given the high proportion of -6mm material in the feed (Honaker et al, 2007)..

The washability table for the feed, product and discard samples is presented in Table 4.27, Figure 4.33 illustrates the relationships between yield and relative density, whilst the yield and ash relationships are depicted in Figure 4.34.

Examination of the feed data in Table 0.27 indicated that an ash content in the order of 14% would be possible at a relative density of 1.45 and would give a yield of approximately 6%. This correlated reasonably well with the product results which produced a 15.3% ash and 7.6% yield at a relative density of 1.40. The higher yield and ashes obtained in the product sample being attributed to the finer and lighter middlings fraction being directed into the product.

Table 4.27: Washability Data

	Feed				Product				Discard			
	Fractional		Cumulative		Fractional		Cumulative		Fractional		Cumulative	
RD	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
1.40	2.4	10.2	2.4	10.2	7.6	15.3	7.6	15.3	0.0	0.0	0.0	0.0
1.50	6.7	20.4	9.1	17.7	5.8	20.1	13.3	17.4	1.0	31.0	1.0	31.0
1.60	11.8	27.3	20.9	23.1	12.8	27.3	26.1	22.2	6.1	41.6	7.1	40.1
1.70	9.5	33.6	30.4	26.4	10.0	34.0	36.1	25.5	1.6	43.8	8.6	40.8
1.75	3.0	37.0	33.4	27.3	3.9	36.0	40.0	26.5	2.1	45.5	10.8	41.7
1.80	3.4	40.0	36.8	28.5	2.8	38.9	42.8	27.3	1.4	47.2	12.1	42.3
S1.80	63.2	70.6	-	-	57.2	67.5	-	-	87.9	75.2	-	-
Total	100.0	55.1	100.0	55.1	100.0	50.3	100.0	50.3	100.0	71.2	100.0	71.2

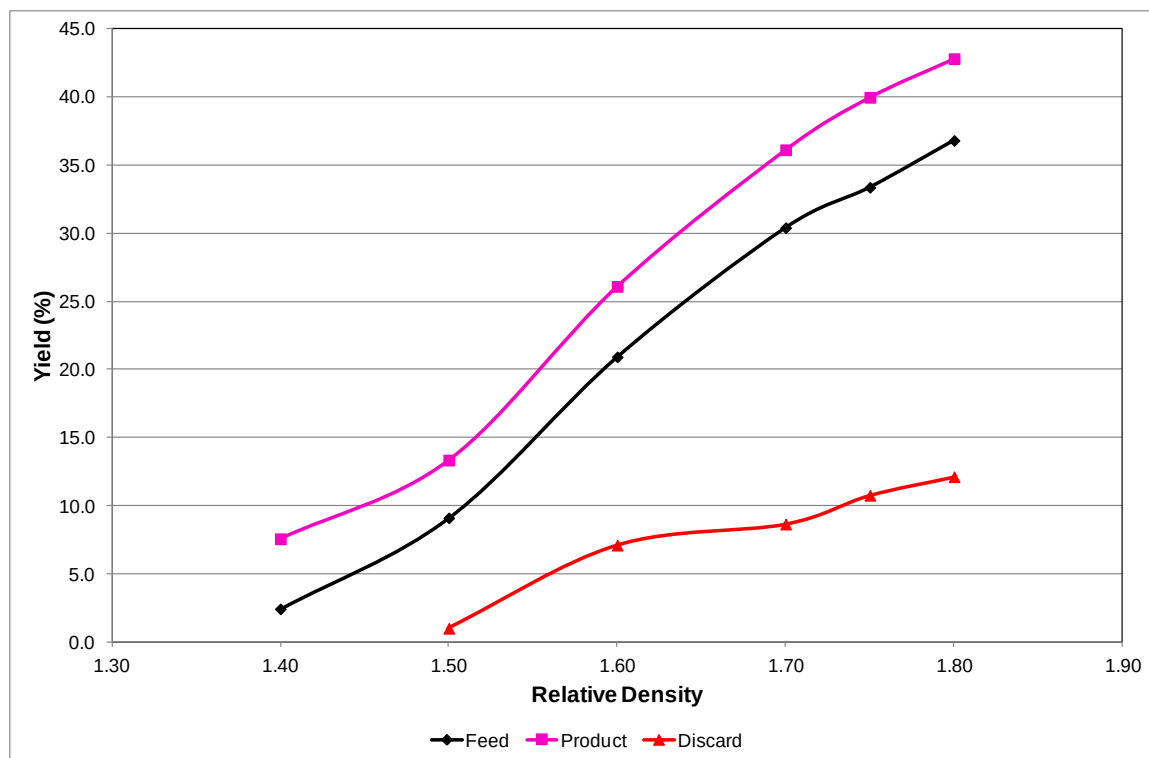


Figure 4.33: Relationship between yield and relative density

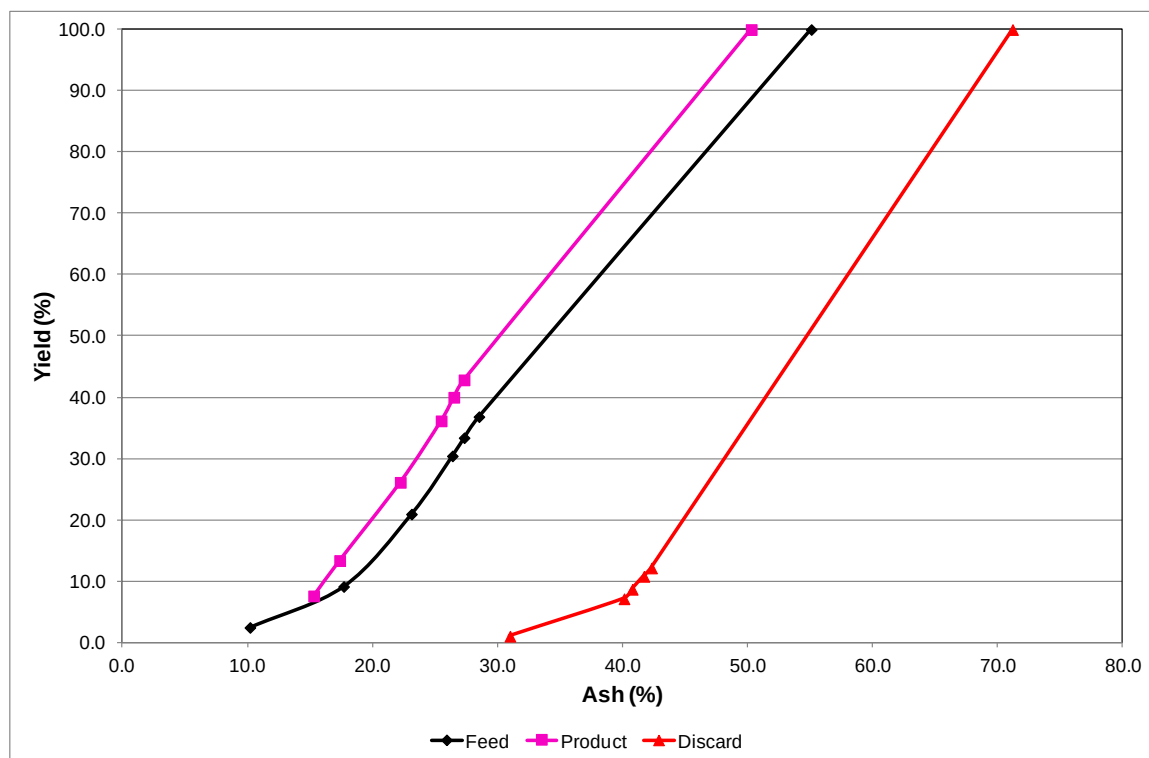


Figure 4.34: Relationship between yield and ash

No material was found to float at a relative density of 1.40 in the discard, and only approximately 12% of the material was noted to float at a relative density of 1.80. This is in line with the findings of both Honaker (2007) and de Korte (2009) where the unit was found to have application when separation densities in the order of 2.0-2.3 are required.

4.4 Potential Flowsheet Options for Destoning

Based on the work conducted on the arising discard from the 1 tonne bulk samples and the 20 tonne bulk sample, a series of flowsheet options were simulated using Excel.

4.4.1 Destoning By Screening

As noted in section 4.3.4 of this chapter, destoning could potentially be effected by means of screening out the high ash containing coarse sized material. Screen efficiency was simulated with the view to reject the +8mm material to oversize, and allow the -8mm to be further processed using dense medium separation.

The simulation was carried out using the Whiten Equation (Equation 4.2) shown below:

$$P = \frac{1}{1 + \exp\{1.0986(d_{50} - d)/Ep\}} \dots\dots\dots \text{Equation 4.2}$$

Where P is the partition coefficient, d_{50} is the separation size and d is the specific size and Ep is the ecart probable moyen. Though traditionally used in dense medium simulations, it was successfully applied by De Korte (2008) and therefore used in this instance. The d_{50} was set at 8mm and an estimated Ep of 0.25, as determined by De Korte (2008) was applied to the fractional screening data. The detailed simulation results are shown in Appendix 5, whilst Table 4.28 presents a summary of the simulations.

Table 4.28: Summary of screen simulations at a d_{50} of 8mm and E_p of 0.25

Sample	Screen Oversize		Screen Undersize	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)
Mutale Shaft	53.5	65.0	46.5	46.9
Mutale Plant Waste	57.5	69.1	42.5	49.8
Vhukati Shaft	63.7	66.8	36.3	49.9
Vhukati Plant Waste	55.7	65.2	44.3	47.7
Goni Plant Waste	52.8	70.0	47.1	47.5
Average	56.7	67.2	43.3	48.4

The screening simulations indicated that a significant upgrade could be achieved by screening at 8mm. Although the product specification could not be met, it is possible that a saleable product could be made from this material if subjected to upgrading by either dense medium or gravity separation techniques.

4.4.2 Destoning by Means of the FGX Separator

Based on the work presented in section 4.3.5, the FGX Separator would not be a suitable method of destoning due to poor upgrading of the material fed to it due to the high proportion of -6mm present. Therefore no further work was considered using this unit, since it was considered unlikely to produce a significant improvement on the test performed.

4.4.3 Destoning by Dense Medium Separation after Screening at 8mm

Section 4.4.1 established that screening at 8mm showed potential to remove the high ash material from the discard, consideration was therefore given to further upgrading of the -8mm by means of dense medium separation, most likely a cyclone given the particle size (-8+0.5mm) to be processed. The circuit flowsheet is depicted in Figure 4.35:

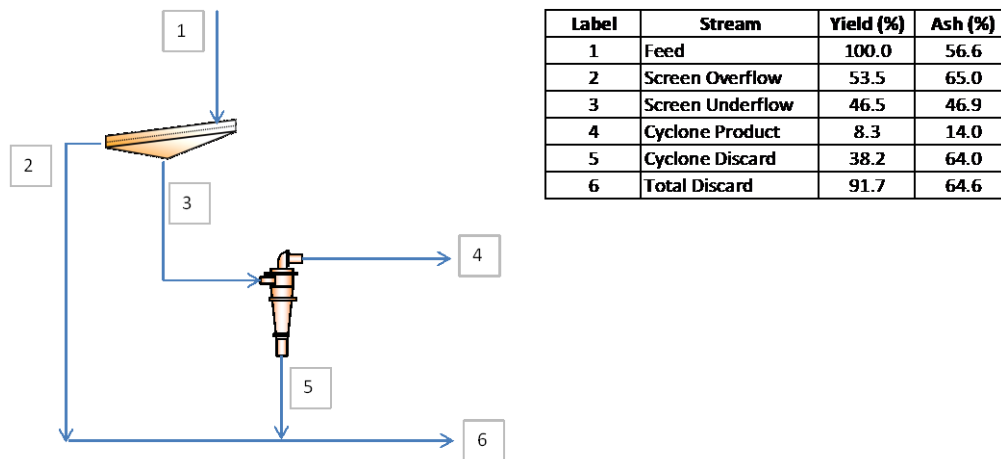


Figure 4.35: Destoning circuit simulation using a combination of screening and dense medium separation for the Mutale Shaft discard sample

Two scenarios were considered, the first targeting a 14% ash product for use in coke making, the second at a higher ash content suitable for power generation by Eskom.

4.4.3.1 Scenario 1: Production of a 14% ash coking coal

The use of dense medium separation on the -8mm material was simulated also using the Whiten equation. The -8mm material for each sample was mathematically combined in mass proportion and the Whiten equation applied using a typical dense medium cyclone E_p of 0.025, whilst the d_{50} was varied to obtain a product specification of 14.0% ash. The detailed results of the simulation are contained in Appendix 5 and summarized in Table 4.29.

Table 4.29: Summary of dense medium simulations on -8mm discard at an E_p of 0.025

Stream	Mutale Shaft		Mutale Plant Waste		Vhukati Shaft		Vhukati Plant Waste		Goni Plant Waste	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
Feed	100.0	56.6	100.0	60.9	100.0	60.7	100.0	57.5	100.0	59.4
Screen Overflow	53.5	65.0	57.5	69.1	63.7	66.8	55.7	65.2	52.8	70.0
Screen Underflow	46.5	46.9	42.5	49.8	36.3	49.9	44.3	47.7	47.2	47.5
Cyclone Product	8.3	14.0	8.2	14.0	6.1	14.0	9.4	14.0	14.2	14.0
Cyclone Discard	38.2	64.0	34.3	59.9	30.2	58.7	34.9	58.2	33.0	63.6
Total Discard	91.7	64.6	91.8	65.7	93.9	64.2	90.6	62.5	85.8	67.5
Cyclone d_{50}	1.486		1.482		1.482		1.491		1.526	

Relative densities in the order of 1.48 to 1.49 would be required for the Mutale and Vhukati samples, whilst a slightly higher operating density in the order of 1.53 would be required for the Goni sample.

The yields achieved were generally low ranging between 6-14.2% of feed to the cyclone. Once this has been worked back to ROM feed, and assuming an average plant yield of 55%, the yield of saleable product would therefore range between approximately 3-6%, which would not be economically viable to justify a dedicated discard retreatment plant. It is also noted that these yields are consistent with the discard losses measured in previous sections, and further confirm the hypothesis that the losses of product to discard was from the finer size fractions.

Based on the vitrinite examination by relative density of the discards shown in Table 2.2 in Chapter 2, at relative densities in the order of 1.50 to 1.55, the destoned product would be expected to contain a vitrinite content in the order of 80%.

An alternative would be to consider retreating the discard through the existing coal preparation plant with suitable modification made to the circuit. Although the capital expense to reconfigure the plant would not be excessive, the operational costs would most likely make this option to be unfeasible.

4.4.3.2 Scenario 2: Production of Eskom power station feedstock

Having seen from the previous section that a low yield of 14% ash material may not be economically viable the logical next step would be to examine the potential for supply to Eskom for power station feed. Simulation of dense medium separation on the -8mm material was performed at a separation relative density of 1.80 and E_p of 0.025. The detailed results of the simulation are contained in Appendix 5 and summarized in Table 4.30.

Table 4.30: Summary of dense medium simulations on -8mm discard at an E_p of 0.025 and separation relative density of 1.80

Stream	Mutale Shaft		Mutale Plant Waste		Vhukati Shaft		Vhukati Plant Waste		Goni Plant Waste	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
Feed	100.0	56.6	100.0	60.9	100.0	60.7	100.0	57.5	100.0	59.4
Screen Overflow	53.5	65.0	57.5	69.1	63.7	66.8	55.7	65.2	52.8	70.0
Screen Underflow	46.5	46.9	42.5	49.8	36.3	49.9	44.3	47.7	47.2	47.5
Cyclone Product	20.2	29.4	20.4	27.0	16.9	29.2	22.3	27.6	22.8	23.9
Cyclone Discard	26.3	71.7	22.1	73.3	19.4	70.4	22.0	70.3	24.4	71.7
Total Discard	79.8	67.2	79.6	70.3	83.1	67.6	77.7	66.6	77.2	70.5

By processing the -8mm at a relative density of 1.80, it was able to achieve yields of 17-23% at ashes ranging between 23-29%, which would potentially be suitable for the production of an Eskom product in terms of ash. A vitrinite content of approximately 60% would be expected at this relative density as suggested by Table 2.2. Again working the product yield back relative to the ROM, the yields were noted to range between 8-10%. Washing at a slightly higher relative density would bring several of the samples in line with a typical Eskom specification of 30% ash, though the gains in yield would not be significant, with the exception of the Goni material. At these yield levels it is also unlikely that this option for the plant is viable, given the lower price of Eskom material relative to prime coking coal. The situation would further be exacerbated once transport of the product to a power station is factored in.

4.5 Chapter Summary

In this section of the work, a literature survey with respect to destoning of coal was undertaken, and from that technologies considered appropriate for the upgrading of the Tshikondeni discards were examined.

Five one tonne bulk samples were collected from the three operating shafts, viz. two from Mutale, two from Vhukati and one from Goni. The ash content of these samples was determined to be between approximately 56-61%.

From the washability by size fraction exercise it was noted that some destoning could be achieved by screening out the +8mm, which contributed to between 53-58% of the mass in the samples at ash contents of between 65-70% ash. The -8mm fractions has ash contents of between approximately 46-50%.

Simulation of a dense medium separation on the -8+0.5mm fraction showed that a coking coal specification of 14% ash could be met at separation relative densities in the order of 1.49-1.53g/cm³, though the mass yields were very low at between 6-14%. A vitrinite content in the order of 79-83% would also be expected from this product.

Given that the average plant yield is typically in the order of 55%, the yield from rewashing this fraction would equate to approximately 3-8% of the run of mine, and is unlikely to produce a return on the processing costs.

A further simulation was performed at a separation relative density of 1.80g/cm^3 , where it was found that product ash contents of between approximately 24-29% could be achieved. The vitrinite content would be expected to be approximately 60%. The mass yields were also low at between 22-26%, which equated to approximately 8-10% on a run of mine basis. Despite the higher volume of material available for sale, the revenue for this product would be substantially less than that for hard coking coal, and the transport costs alone would outweigh the costs of production since it would have to be trucked approximately 500km to the nearest power station.

It was found that an air jig such as the FGX Separator would not be suitable for the discard at Tshikondeni due to the high proportion of -6mm in the material; however, dense medium separation would be very applicable. Should this have worked, it would have been an ideal process due to the severe water scarcity in the Tshikondeni area, which has approximately 300mm of rainfall per annum based on the historical records from the Tshikondeni Survey department.

The use of the Reflux classifier to treat the fines in the range -1.4+0.6mm was not considered due to the short life of mine by the time the unit had been fully commissioned at Exxaro Research and Development (late 2012).

Overall, it was noted that destoning could be effected by the application of dense medium, with or without the use of screening. From this section of the work, the destoned product could be milled to create liberation of the coal from the waste rock, and thereafter froth flotation employed to produce the hard coking coal product desired by ArcelorMittal South Africa.

Liberation and milling are considered in the next chapter.

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Chapter 5: Liberation and Milling

This chapter begins with a high level and general literature survey that covers liberation of values from waste rock and then moves into modelling theory as applied to milling. The latter acts as the background for the reader to understand the experimental procedures employed by Mintek, where the test work was performed, which is also described in detail.

Representative sub samples of the five samples (1 tonne bulk samples taken from the three shafts) were subjected to the Mintek laboratory batch ball grind mill test procedure, and circuit simulations performed to produce a mill product of 80% passing 212 μ m, the anticipated size for the feed into the froth flotation circuit after milling the destoned discards.

Specific energy consumptions for the five samples were determined to be between approximately 6-7kWh/t, with circulating loads of between approximately 121-147%.

5.1 Literature Survey

5.1.1 Liberation

Due to its importance, liberation is a concept that is described in the early sections of most minerals processing text books such as: Wills and Napier-Munn (2006), Kelly and Spottiswood (1982) and Woollacott and Eric (1994). All of these texts express that one of the major objectives of liberation is the release of the valuable mineral from the associated waste (gangue) rock at the coarsest possible particle size. High grade solid products would therefore require higher degrees or level of liberation. The degree of liberation is defined as the percentage of the required mineral occurring as free particles in the ore in relation to the total content. This will be higher in cases where there are weak boundaries between the ore and the gangue particles. Unfortunately this only occurs in rare instances, and more often than not, it is adhesion between the mineral and gangue is strong and, during comminution, the various constituents are cleft across.

Figure 5.1, taken from Wills and Napier-Munn (2006), shows a cross section through a typical ore particle and well illustrates the dilemma frequently faced by the minerals processing engineer. Within the ore fragment, the “A” represents the valuable mineral, whilst “AA” represents a region rich in the valuable mineral, but is highly intergrown with the gangue mineral. Breaking down the ore fragment into smaller pieces will produce a range of products from fully liberated ore and gangue particles along with the four types of ore/gangue mixed particles illustrated. Type 1 particles are noted to be rich in the mineral and would be considered to have an acceptable level of gangue locked to the mineral, though the overall concentrate grade would be limited. Type 4 particles would be considered as tailings and contain an acceptable level of mineral loss. Type 2 and 3 particles would be called “middlings” and how they contact each other can be an important factor in determining the particles’ behaviour in a separator.

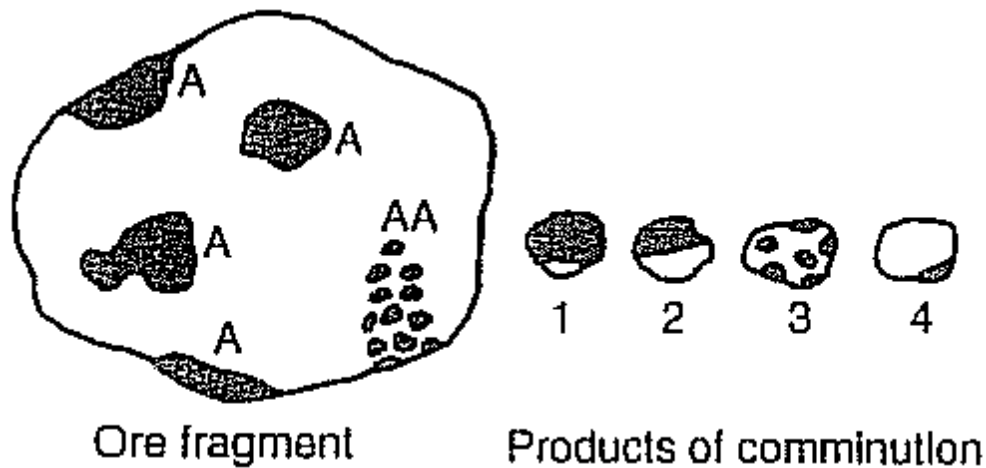


Figure 5.1: Cross section of ore particles (Wills and Napier-Munn, 2006)

Kelly and Spottiswood (1982) further define middlings particles into four broad categories as illustrated in Figure 5.2. Particles of Type I have the mineral and gangue making contact along a rectilinear boundary, and indicates that the grain size is considerable larger than the particle size. Further size reduction would readily improve the liberation of the valuable mineral from the gangue in this case. Type II particles show that the mineral is in the form of veins within the particle. It is evident that liberation here would be very difficult to achieve. Particles of Type III are composed of a shell of gangue surrounding the desired mineral. Further size reduction will result in some liberation of the mineral from the gangue shell, but a significant proportion of difficult to treat particles will remain. Type IV shows that the particles contains mineral occlusions within a gangue matrix. Liberation of the values from this particle would quite obviously be very difficult.

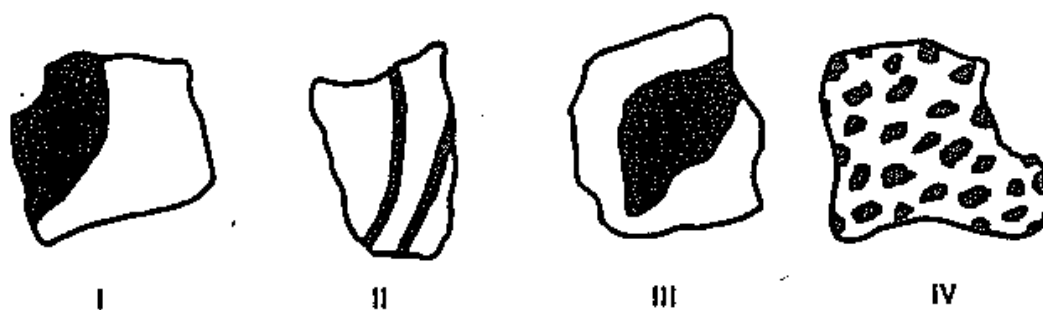


Figure 5.2: The four basic types of middling or locked particles (Kelly and Spottiswood, 1982)

Kelly and Spottiswood (1982) also state that minerals processing separators exploit a specific property of the particles in question, and that how the separator “sees” the particle is important. In froth flotation, a certain proportion of the particles surface should be floatable. In magnetic separation, any magnetically susceptible particle will be recovered, even if covered in a shell similar to a Type III particle in Figure 5.2. In gravity separation, the particle will have an apparent density directly proportional to its volumetric composition, and thus a comparatively wide spectrum of behaviour.

Having considered that the discard generated by the beneficiation plant at Tshikondeni has been destoned, it is therefore necessary to re-treat the floats product by milling to liberate any remaining coking coal from the true waste rock. Furthermore, the plant also uses a flotation section processing -1.4+0mm and this would be source of losses. The use of a mill would therefore reduce the size of particles too heavy to be lifted into the flotation product and also generate fresh surfaces to overcome any surface tarnishing that could have occurred.

5.1.2 Milling Modelling Theory

The steady-state and dynamic performance of grinding mills can be described using a population balance approach to quantify the size distribution and composition of material inside the mill. It is common to select sizes conforming to a root-two geometric progression given by x_i , where the subscript i serves as an index to reference each size class. The first or largest size x_1 , is normally selected to be larger than the largest particles likely to be encountered in the feed stream. If there are n size classes, x_n references all particles with sizes between zero and x_n . The composition of material in each size class can be expressed in terms of rock type, assay value and mineral content. Consideration should also be given to the state of liberation of each species. Breakage behaviour inside the mill is also usually quantified in terms of a specific breakage rate function and separate breakage distribution function.

The specific breakage rate function for tumbling and stirred media mills, $S_{i,k}$, can be defined as the statistical average of the fractional rate (per unit time) that particles of composition or species type k that break out of size class i . Because a time-based breakage rate function can be sensitive to mill geometry and operating conditions, it

is desirable to make a transformation to an energy-normalised breakage rate function defined by $S_{i,k}^E = S_{i,k}M/P$ (Herbst and Fuerstenau, 1973), where P is the net mill power and M is the total mass of the material to be broken in the mill. The energy-normalised breakage function is generally insensitive to scale up. In principle, it should be possible to measure the specific breakage rate function directly by using radioactive tracer techniques to “tag-and-track” each species of interest during its passage through the mill. This is very difficult and impractical to do on a routine basis. Consequently, simplifying assumptions must be made to obtain an estimate of the breakage rate function. A commonly made assumption is that the breakage rate function can be represented by a simple equation with parameters that can be back-calculated from test data (Austin, Klimpel and Luckie, 1984). The equation takes the form:

$$S_i^E = \kappa \chi_i^\alpha \frac{1}{1 + (\chi_i / \mu)^\lambda} \dots \dots \dots \text{Equation 5.1}$$

Where κ , α , χ and μ parameters of the selection function (specific breakage rate) and χ_i is the particles size (mm).

The primary breakage distribution function for a tumbling or stirred media mill, $b_{i,j,k}$ is defined as the mass fraction of species k breaking out of size class j that appears in size class i before particles have had a chance to be re-broken. Again, in principle, it should be possible to estimate the breakage distribution function using radioactive tracer techniques, but such an approach has yet to be proven viable. To resolve this dilemma, it is possible to adopt a similar approach to that described above by assuming that the breakage distribution function can be represented by a simple equation with parameters that can be identified by back-calculation. For ball milling, the cumulative breakage distribution function B_{ij} (from which b_{ij} can be evaluated as $b_{ij} = B_{ij} - B_{i+1,j}$) takes the form:

$$\begin{aligned} B_{ij} &= \Phi_j (\chi_{i-1} / \chi_j)^\gamma + (1 - \Phi_j) (\chi_{i-1} / \chi_j)^\beta, \quad i \geq j \\ \Phi_i &= \Phi_{i-1} (\chi_i / \chi_{i-1})^{-\delta}, \quad i = 2, 3, \dots, n-1, \quad \Phi_{i-1} (\chi_i / \chi_{i-1})^{-\delta} < 1 \\ &= 1, \quad \Phi_{i-1} (\chi_i / \chi_{i-1})^{-\delta} \geq 1 \end{aligned} \quad \text{Equation 5.2}$$

Where ϕ , γ and β are the parameters breakage distribution function as described by Austin, Klimpel and Luckie (1984). The above mentioned models/equations will be used to fit the batch ball milling results.

A balance of material broken into and being broken out of size interval i is:

Rate of accumulation of size i material = production from breakage of all larger sizes ($j=1$ to $i-1$) – rate of breakage of size i to smaller sizes.

Symbolically this is given by the two following equations as proposed by Austin, Klimpel and Luckie (1984):

$$\frac{dw_i(E)W}{dE} = -S^E_i w_i(E)W + \sum_{j=1}^{i-1} b_{ij} S^E_j w_j(E)W \quad \text{.....Equation 5.3}$$

or

$$\frac{dw_i(E)}{dE} = -S^E_i w_i(E) + \sum_{j=1}^{i-1} b_{ij} S^E_j w_j(E) \quad \text{.....Equation 5.4}$$

The use of analytical functions ensures that the predicted size distributions are constrained to physically realizable values.

5.2 Laboratory Batch Milling Test Procedures

Representative samples of the destoned discard samples from the three shafts as discussed in Section 4.2.3 were sent to Mintek in Johannesburg for batch milling testwork to assist in the sizing of the mill required ahead of the flotation process.

The milling test sample was screened at 0.5mm, the +0.5mm fraction was destoned by a process of float and sinks at a single relative density fraction of 1.75. This slight change in relative density was intended to give an overall improvement of the quality of the destoned fraction and should produce a better flotation product compared to previous tests performed to this point. The material that floated at RD=1.75 was combined with the -0.5mm screen fraction as illustrated in Figure 5.3.

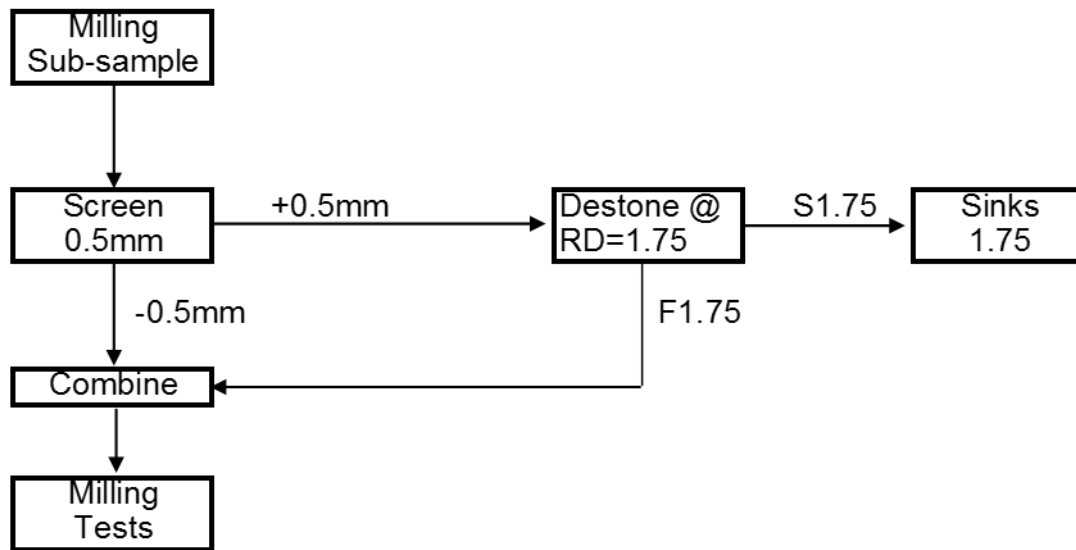


Figure 5.3: Sample preparation prior to milling testwork

5.2.1 Ball Milling Facility and Milling Conditions

The batch ball milling setup at Mintek consists of a 265 × 405 mm vessel as shown in Figure 5.4 below. Figure 5.5 shows the detail of the load cell and tachometer used to monitor mill speed and mechanical torque on the mills. The sensor outputs are interfaced to a computer to permit real-time estimation of net power consumption. Table 5.1 and Table 5.2 give the details of the milling conditions used.

Tests were conducted to determine the relationship between specific energy input and the size distribution of the milled product under the given conditions.



Figure 5.4: Mintek 265mm diameter grind mill



Figure 5.5: Tachometer and load cell for real time measurement of net power

Table 5.1: Ball milling conditions used

Parameter	Ghoni Plant Waste	Mutale Plant Waste	Mutale Shaft	Vhukati Shaft	Vhukati Plant Waste
% grinding media in mill	35	35	35	35	35.0
Specific gravity of balls	7.85	7.85	7.85	7.85	7.85
mill diameter (inside liners (m))	0.265	0.265	0.265	0.265	0.265
mill length (inside liners (m))	0.405	0.405	0.405	0.405	0.405
Specific gravity of ore	1.6288	1.6362	1.642	1.703	1.643
% filling of voids with pulp	100	100	100	100	100
% solids by mass of pulp	57.133	57.24	57.33	58.21	57.34
J_{trunnion} (fraction)	0.35	0.35	0.35	0.35	0.35
ball charge mass (g)	36824	36824	36824	36824	36824
total charge volume (m^3)	0.0078	0.00782	0.00782	0.00782	0.007818
volume of slurry (m^3)	0.00313	0.00313	0.00313	0.00313	0.00313
mass of solids in mill (kg)	2.29	2.30	2.311	2.396	2.312
volume of water (litres)	1.7	1.72	1.72	1.72	1.72
slurry density (sg)	1.28	1.29	1.29	1.32	1.29
% solids by volume	45	45	45	45	45

Table 5.2: Ball size distribution

Ball size (mm)	Cum %	Size %	Mass (kg)
70	100.00	46.33	17.061
60	53.67	28.13	10.360
50	25.53	15.47	5.697
40	10.06	7.34	2.702
30	2.73	2.73	1.004
20	0.00	0.00	0.000
Total	-	100.00	36.824

5.2.2 Continuous Circuit Simulations

Computer simulations were conducted to establish the mill net power (kW) required to achieve 80% passing 212 microns in the circuit product at a feed tonnage of 1 t/h, for all five samples tested.

For batch milling, the residence time distribution corresponds to plug flow conditions. However, for most plant mills fed continuously, the residence time distribution can be best modelled as three fully mixed tanks in series, one large and two small as depicted in Figure 5.6. As a rule of thumb, three tanks in series can provide a good approximation of residence time distributions likely to be encountered in practice.

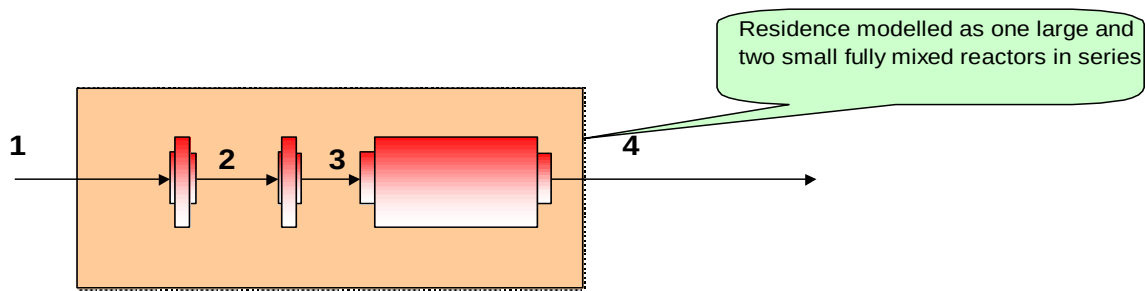


Figure 5.6: Modelling residence time distribution of production mill

The performance of the hydrocyclone closing the milling circuit was simulated in terms of a simple partition function with parameters the fines bypass, cut-size, and the sharpness of cut described by Plitt (1976). The equation is of the form:

$$R(d) = R_f + (1 - R_f) \left(1 - \exp(-0.693(d / d_{50c})^m) \right) \quad \text{.....Equation 5.5}$$

Where $R(d)$ is the fractional recovery of particles of size d to the underflow or partition number. The parameters are R_f , d_{50c} and m are the fines bypass, cut size, and sharpness of separation, respectively. The R_f is equivalent to the amount of water that reports to the underflow.

5.3 Laboratory Batch Milling Test Results

5.3.1 Laboratory Batch Ball Grind Milling Results

The measured and model particle size distributions (PSD) are illustrated in Figure 5.7 to Figure 5.11 respectively, which represented the modelled and measured flow rates using the first order rate hypothesis describing the breakage of the five samples tested. The values of the parameters obtained for modelling using the least squares fit are provided in Appendix 5.

From Figure 5.7 to Figure 5.11 it was observed that the first order rate hypothesis was suitable to describe the breakage of the five materials, since the model fits the measured data to a high degree. This then implied that mass balances for

continuous operation can be carried out using the evaluated parameters of breakage and selection functions obtained from the batch grind mill test data.

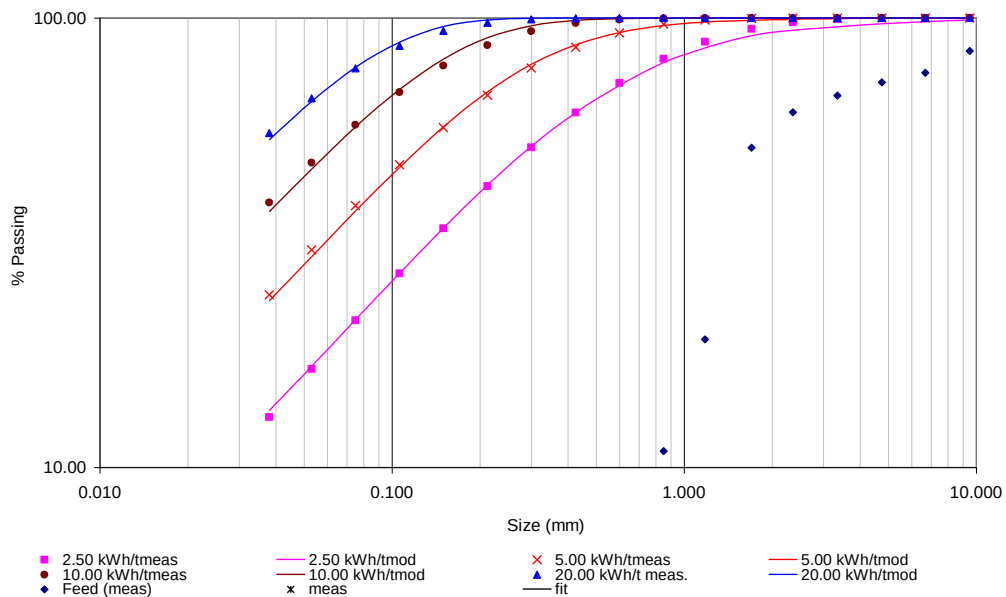


Figure 5.7: Goni Plant Waste – measured and model PSD

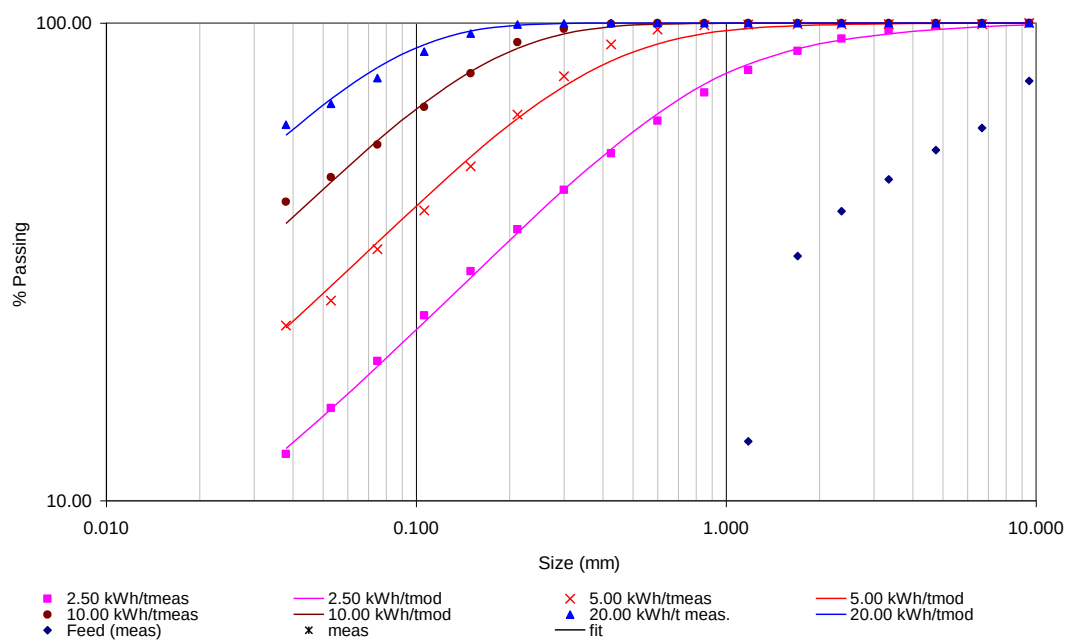


Figure 5.8: Mutale Plant Waste – measured and model PSD

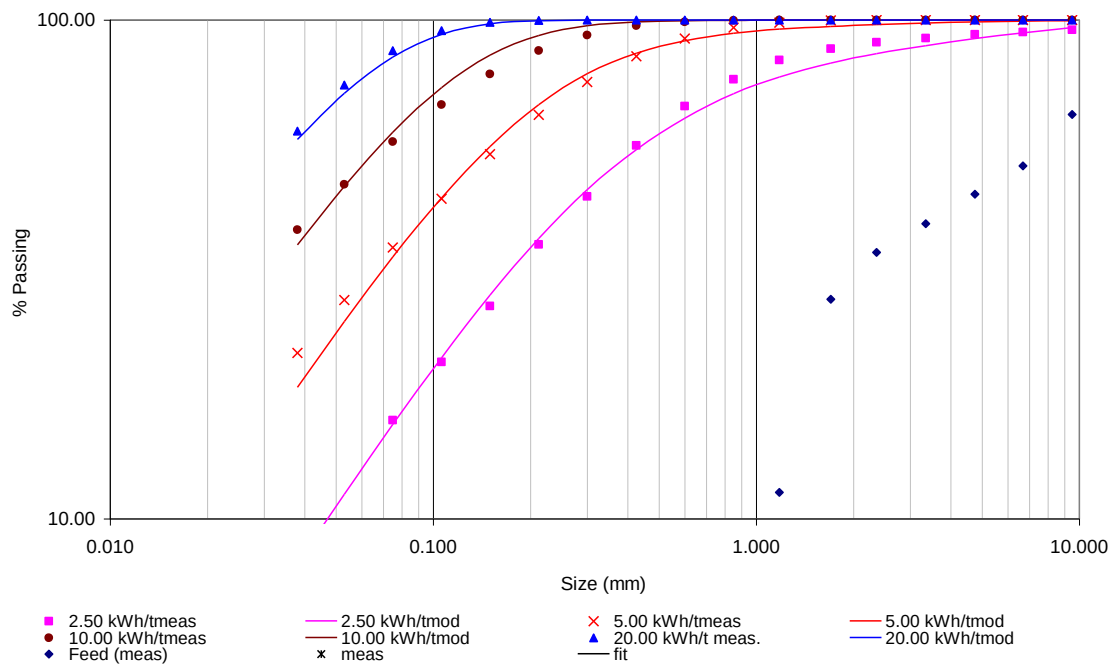


Figure 5.9: Mutale Shaft Waste – measured and model PSD

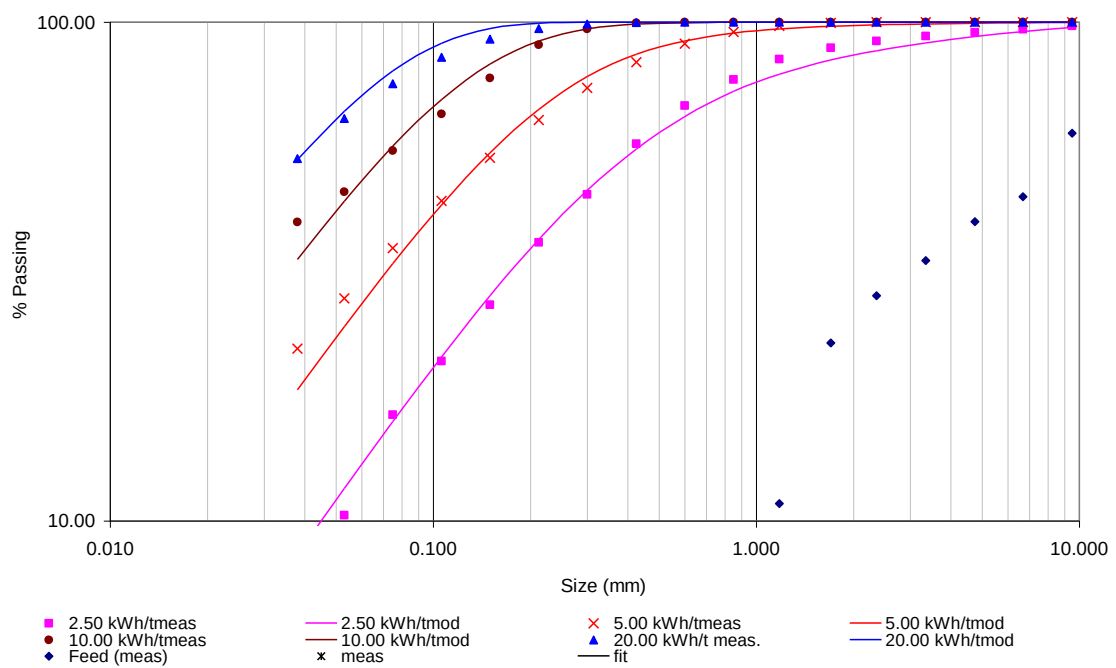


Figure 5.10: Vhukati Shaft – measured and model PSD

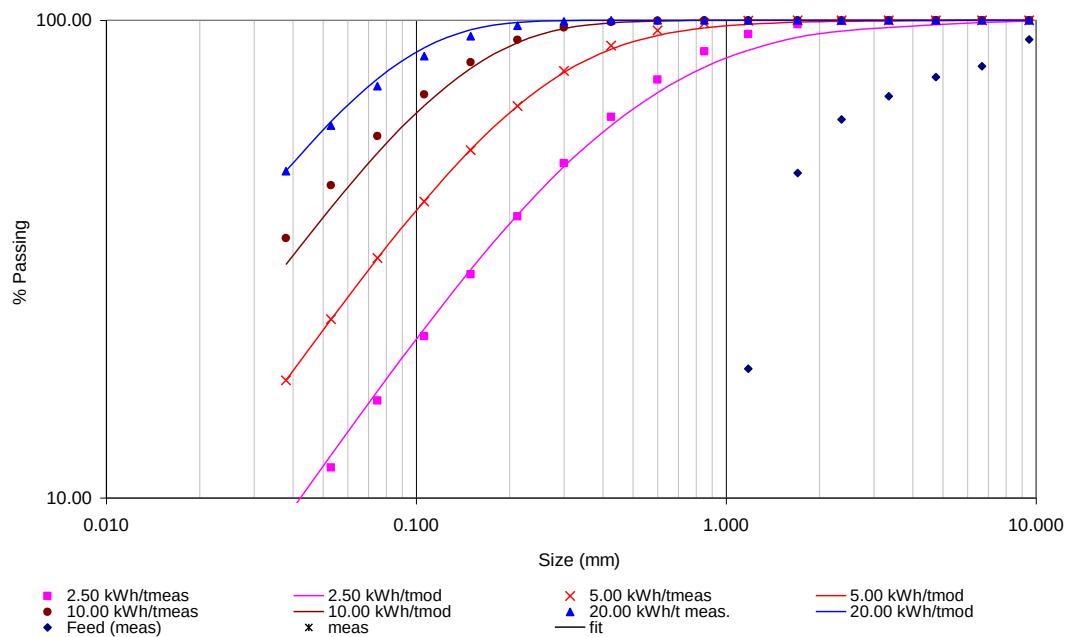


Figure 5.11: Vhukati Plant Waste – measured and model PSD

The straight line trend observed at the finer particle sizes indicated that under the milling conditions described in 5.2.1 above, that the particles are readily comminuted. Changes in ball size distribution could change the selection function values considerably, i.e. the slope and shape of the curves. If the same milling conditions are maintained, but a smaller ball top size used, this may result in a downward shift in the curve indicating slower breakage kinetics, which would in turn mean that more power will be required to reach the same grind.

At larger particle sizes it is observed that the rate of breakage slows down considerably and becomes constant with increasing particle size. This trend is observed with all five of the samples tested. Any changes in the laboratory parameters used would therefore most probably result in changes in the selection function, thereby allowing the same conditions to be used on large scale ball milling.

The breakage rates shown in Figure 5.12 indicated that the five samples tested have similarities in their breakage characteristics suggested by the similarities in terms of slopes and shapes of the graphs plotted.

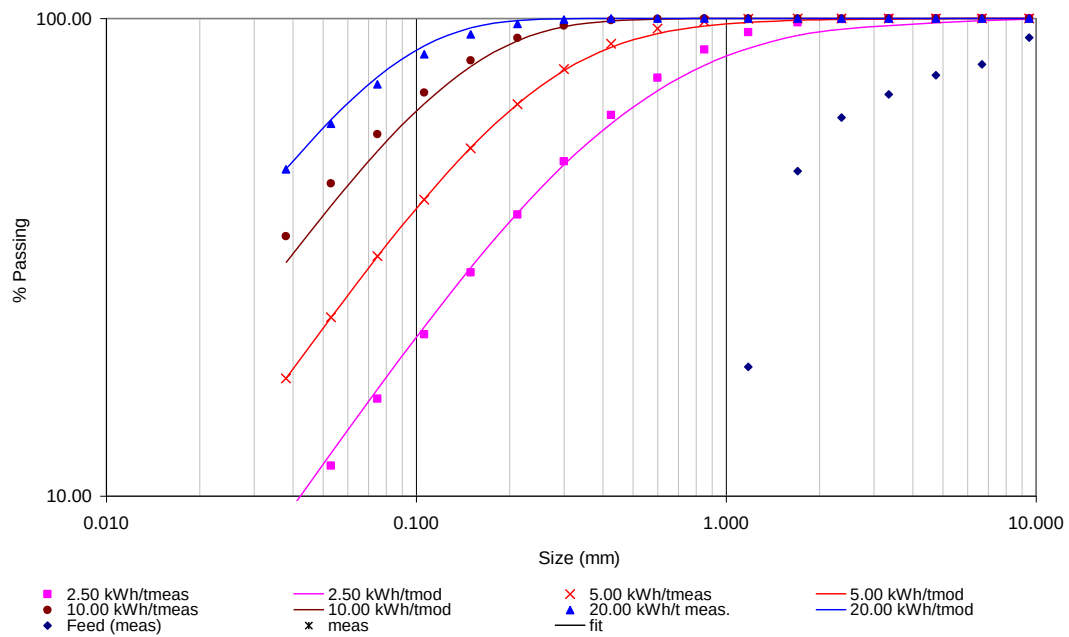


Figure 5.12: Breakage rates

5.3.2 Continuous Closed Ball Milling Circuit Simulation

Appendix 3 also contains detailed mass balances and particle size distributions for the evaluation of the circulation loads and specific energy consumption for the closed ball milling circuits as depicted in Figure 5.13. The results of the simulation conducted on a fresh feed tonnage (stream 1) of one tonne per hour are summarized in Table 5.3 for all five samples.

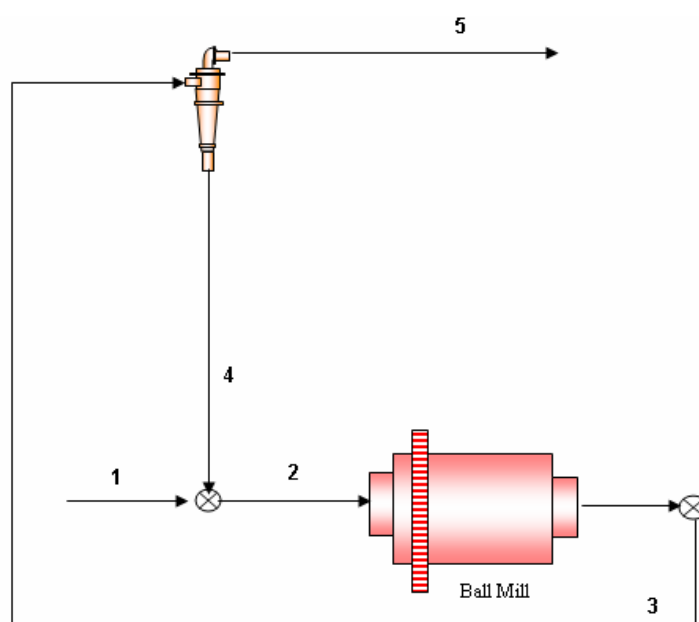


Figure 5.13: Circuit flowsheet

Table 5.3: Overall mass flow based on 1 t/h new mill feed

Sample	Stream 1	Stream 2	Stream 3	Stream 4	Stream 5
	New feed	Mill inlet	Mill discharge	Cyc 1. u/f	Cyc 1. o/f
Goni Plant Waste	1.000	2.231	2.231	1.231	1.000
Mutale Plant Waste	1.000	2.179	2.179	1.179	1.000
Mutale Shaft	1.000	2.469	2.469	1.469	1.000
Vhukati Shaft	1.000	2.334	2.334	1.334	1.000
Vhukati Plant Waste	1.000	2.214	2.214	1.214	1.000

The cyclone parameters are presented in Table 5.4 whilst Table 5.5 summarized the required mill power and expected circulating loads for a grind of 80% passing 212 μ m.

Table 5.4: Cyclone parameters

Parameter	Goni Plant Waste	Mutale Plant Waste	Mutale Shaft	Vhukati Shaft	Vhukati Plant Waste
R_f	0.184	0.110	0.156	0.087	0.158
D_{50c} (mm)	0.229	0.204	0.202	0.193	0.209
m	1.209	1.082	1.097	1.081	1.100

Table 5.5: Required mill power and circulating loads for 80% passing 212 μ m

	Goni Plant Waste	Mutale Plant Waste	Mutale Shaft	Vhukati Shaft	Vhukati Plant Waste
Specific energy (kWh/t)	6.27	6.99	6.33	6.22	6.86
Circulating loads (%)	123.00	118.00	146.70	133.00	121.00
kWh per tonne of fresh feed	1.00	1.00	1.00	1.00	1.00

The specific energy consumption indicated that the samples tested are fairly similar in their power requirements for obtaining the desired grind of 80% passing 212 μ m material. The Vhukati Shaft sample is observed to be the sample consuming the least amount of specific energy for comminution with a demand of 6.22kWh/t at a circulating load of 133%. While the Mutale Plant waste has the highest specific energy consumption in comparison to the other four samples tested, it would require an energy input of 6.99 kWh/t with a ball mill circulating load of 118% to achieve the same grind. Figure 5.14 illustrates the effect of the circulating load on the specific energy consumption.

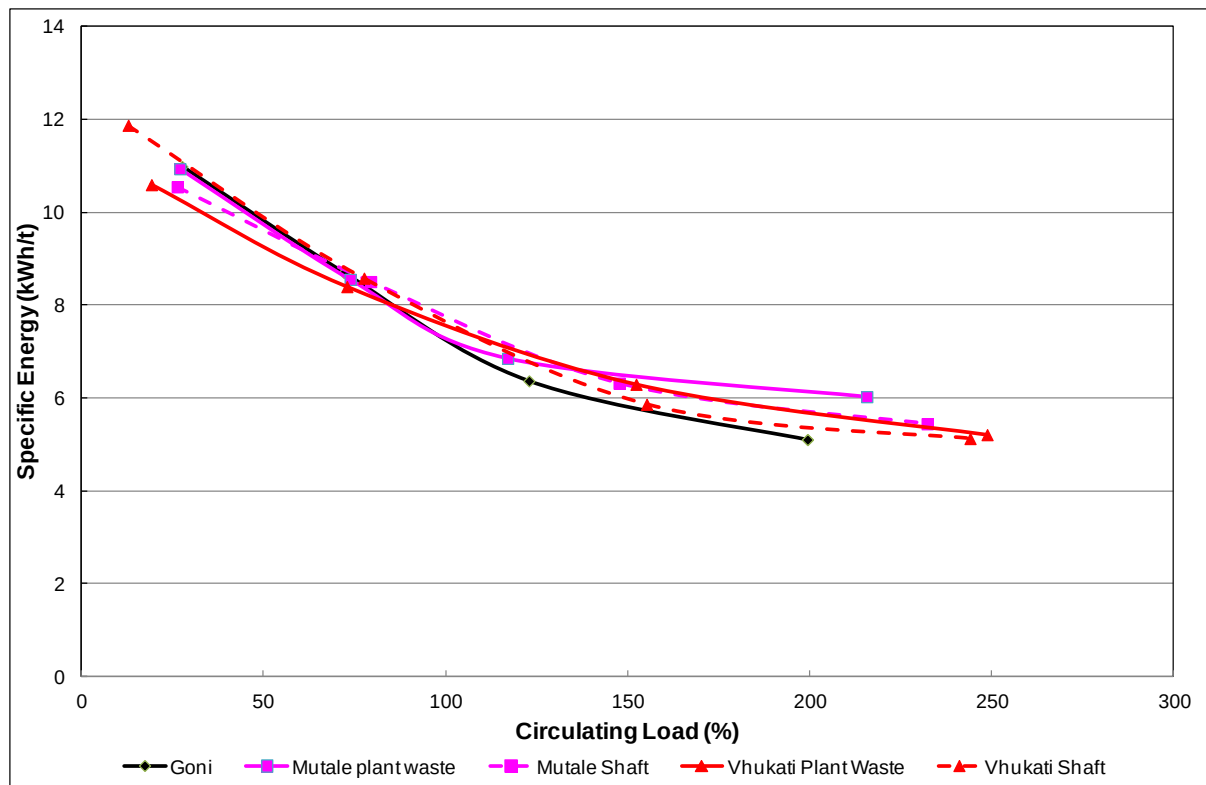


Figure 5.14: Influence of circulating load on power requirements

It can be seen in Figure 5.14 that an increase in circulating loads will decrease the power demand of the circuits and that all of the five samples tested followed the same general trend. As would be expected, any change in circulating load will also affect the performance of the hydrocyclone, and therefore a balance must be maintained between operating the cyclone efficiently and operating an energy efficient circuit. There are several inefficiencies that can occur when operating the circuit at excessively high loads, and it is not uncommon for the pump or the cyclone to become the capacity bottle neck due to volume constraints. These volume constraints are not always apparent and lead to the assumption that increased load has limited the ball mill capacity.

5.4 Chapter Summary

The five sub-samples from the destoned discards from the three operating shafts were subjected to the Mintek laboratory scale batch ball mill test procedure which included a process simulation.

From the testwork conducted it was found that the five samples tested were amenable to milling to 80% passing 212 μ m and would be achieved at specific energy inputs of between 6.22-6.99kWh/t, whilst the circulating loads would be expected to vary between 118-133%.

The energy input is low relative to many minerals as indicated in the standard minerals processing text books (Kelly and Spottiswood, 1982; Wills and Napier-Munn, 2006), and the Tshikondeni discard after destoning would therefore be expected to produce a feed for froth flotation, (which is discussed in the next chapter), without much difficulty.

5.5 Chapter References

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Chapter 6: Froth Flotation

The two previous chapters established that the plant discards can be destoned by either screening, dense medium separation or a combination of screening and dense medium separation, and that the material can be milled with a relatively low specific energy input to 80% passing 212 μ m. This chapter explores the potential of upgrading the destoned and milled discard to a saleable product using froth flotation.

The chapter begins with a literature survey that covers the flotation of fine coal, reagents used in coal flotation and cell design and dynamics. Thereafter, the experimental details for the extensive test work that was undertaken is described. Tests were performed on samples sourced from:

1. The borehole samples taken from the two discard dumps, including the disused return water dam (the Visgat Dam).
2. Current arising discard taken from the monthly composite samples.
3. The 20 tonne bulk sample taken from the old discard dam

It was noted that the use of organic liquids in the destoning process appeared to influence the flotation performance: i.e. lower product qualities and minimal response in terms of selectivity were observed. A short study comparing the use of organic liquids to a suspension of magnetite and water was performed to establish the difference.

The majority of the tests were performed using the standard Tshikondeni reagent suite comprising 85% collector (paraffin) and 15% frother (Betafroth BTN3) which was dosed at a relatively low dosage equating to approximately 0.7kg/t in the flotation circuit. Some modification of the relative density of separation in the destoning stage and reagent suite in the flotation stage were however considered to improve the overall yields and ashes achieved.

Flotation kinetic modelling was performed which indicated that the samples were of a slow floating nature.

6.1 Literature Survey

Wills and Napier-Munn (2006) state that froth flotation is the most important and versatile mineral processing technique, and over the years has continued to be expanded in terms of use, application and processing of larger tonnages. It was originally patented in 1906 to permit the processing of low grade and complex ores, particularly the copper, zinc and lead sulphides, which otherwise would have been considered uneconomic to mine.

Froth Flotation is a physico-chemical separation process that exploits the differences between the surface properties of the valuable minerals and the unwanted gangue minerals. Flotation theory is a very complex subject due to the nature of the interactions and sub-processes between the three phases (solids, water and air) involved.

The desired mineral is recovered from the feed pulp by the following three mechanisms:

- i. True flotation, i.e. selective attachment of the mineral to the air bubbles.
- ii. Entrainment in the water passing through the froth.
- iii. Physical entrapment between particles in the froth attached to air bubbles, which is commonly known as aggregation.

True flotation is generally the dominant mechanism of recovery, though the separation efficiency is also dependent on the degree of entrainment and entrapment.

6.1.1 Flotation of Fine Coal

Froth flotation is widely recognized as the most effective method to separate fine coal from minerals (Allum and Whelan, 1954). This is due to coal being naturally hydrophobic and will therefore tend to strive for air, or any material less polar than water present in a pulp. Hence, a coal particle can easily be wetted by non-polar materials introduced into the slurry, (such as a hydrocarbon oil), rather than be

wetted by water (Briscoe and Vander Veen, 1990). Shales, mudstones and clays have a polar nature, and therefore are preferentially wetted by water.

Within the context of the South African coal industry it is employed on a limited scale, with Tshikondeni being one of the few plants to successfully employ the process. Furthermore, the Tshikondeni plant processes material in the size range $-1.4+0\text{mm}$, which is significantly coarser than would be applied in the Witbank area which typically processes material less than $150\mu\text{m}$.

Horsfall (1992) states that although froth flotation is not density based, the surface activity is governed to a major extent by the mineral content. Horsfall et al (1986) pointed out that the essential problem with flotation of South African coal is its lack of selectivity. A $-500+150\mu\text{m}$ sample of coal was separated first by float and sink analysis into a series of density fractions. These fractions were then floated under identical conditions, and the results were almost independent of the ratio of coal to mineral matter. It should be borne in mind that this work was carried out on material that would under current preparation practice be treated in a spiral and in a bench scale conventional cell, i.e. it was not tested at plant scale. A further problem is that the density of feed coal is approximately 1.55g/cm^3 , which is much lower than for other minerals e.g. 2.7g/cm^3 . Thus coal solids occupy a greater percentage of the volume of a flotation cell than observed in any other mineral flotation process, whilst the equipment used for coal froth flotation is generally the same as that for mineral flotation. This is surprising, as in most mineral separation processes, the froth yield is very low i.e. 1-5%, compared to very high yields seen for coal which can range between 50-90%.

In coal flotation, the hydrophobic coal particle attaches to the air bubbles, whilst the hydrophilic mineral component remains behind in the tailings suspension. Hindermarch and Waters (1951) reported that three surface types can be differentiated:

- i. A hydrophobic surface which is naturally non-wettable in water.
- ii. Polar hydrophilic surfaces.

- iii. Surfaces which are heteropolar. Coal falls into this latter category since it possesses a hydrophobic carbon skeleton as well as hydrophilic surface functional groups.

The difference between a hydrophobic and hydrophilic surface is well illustrated in Figure 6.1 (Woollacott and Eric, 1994) and is determined by the contact angle, where a larger contact angle indicates a hydrophobic surface and vice versa.

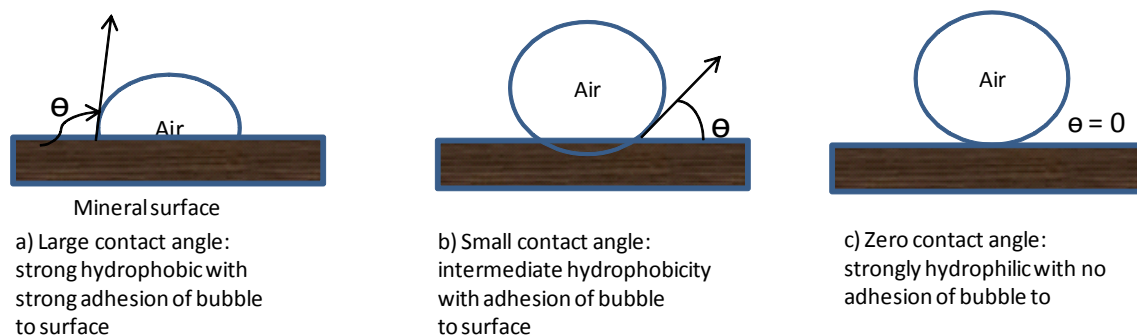


Figure 6.1: Hydrophobicity and contact angle (Woollacott and Eric, 1994)

Several factors influence the natural hydrophobicity of coal (Bunt, 1997). Each of the factors are listed below but are not covered in detail. However, where appropriate for Tshikondeni, some have been expanded upon due to direct relevance.

- i. Coal rank. Tshikondeni is a high rank coal as seen from the typical specifications given earlier.
- ii. Natural floatability. Although coal is generally hydrophobic in nature, the natural floatability in water with respect to the origin and rank is variable (Fuerstenau and Pradip, 1992). The ease of floatability decreases as the rank decreases due to both a decrease in carbon content and an increase in oxygen content (Horsfall, 1992)
- iii. Surface functional group composition. The presence of oxygen containing surface functional groups causes the adsorption of water molecules onto the coal surface. A negative surface charge occurs on the surface and is primarily due to the presence of carboxylic (COOH^-) and phenol (OH^-)

functional groups along with mineral ions on the coal surface (Fuerstenau and Pradip, 1992).

- iv. Degree of oxidation. An increase in the surface functional groups occurs as the degree of oxidation increases leading to a decrease in floatability (Sun, 1954). The oxidation susceptibility of coal decrease with an increase in rank (Taylor et al, 1981), implying that material from Tshikondeni should not be as severely affected as other coal sources in South Africa.
- v. Slime coating and entrainment. Slime is defined as ultra-fine particles that indefinitely remain in suspension, the presence of which during flotation leads to an increase in reagents and a decrease in recovery. Entrainment occurs when fine hydrophilic particles are transported into the froth without being attached to air bubbles (Powell, 1999).
- vi. Particle size distribution. Sun and Zimmerman (1950) reported that coarser particles often require more than one air bubble to float. Turbulence within a cell can also impart low floatability to larger sized particles. Wheeler and Keys (1986) postulated that the floatable particle size limits should be considered as a function of both rank and hydrophobicity, i.e. a higher rank coal particle will float more easily than a lower rank coal for any given size class. This would therefore explain to some extent why Tshikondeni is able to float particles of up to 1.4mm with some degree of success.
- vii. Flotation reagents. Discussed in more detail in section 6.1.2 below.
- viii. Petrographic composition of the coal. The floatability of coal increases with rank due to the lower oxygen content (Ye et al. 1989), whilst Barnwal et al (2000) reported that in terms of lower collector dosages that the flotation rates for the different macerals was observed to be in the order vitrinite, then lipitinite, and finally inertinite. The coal mined at Tshikondeni fits with Barnwal et al's observation in that low collector dosages are used (~0.7kg/tonne) and contains a very high proportion of vitrinite relative inertinite. Liptinite is typically not present in the Tshikondeni material.
- ix. Pulp pH
- x. Pulp density and temperature

6.1.2 Reagents Used in Coal Flotation

The correct selection of reagents for coal flotation is important since the relative density of coal is close to that of water compared to most minerals, which creates an almost neutral buoyancy and difficulty in attaching to bubbles: i.e. the particles tend to follow the streamlines around the bubbles rather than making contact and attachment to the bubble (Powell, 1999). Larger particles are also subjected to gravitational forces (or centrifugal forces if pass around a bubble) that exceeds the binding energy to the bubble and hence loss of these particles to the tailings stream.

Laskowski (1996) summarized in tabular form the common reagents applied in coal flotation as presented in Table 6.1.

Table 6.1: Coal flotation reagents (Laskowski 1996)

Type	Use	Functional Group	Examples	Action
Non-polar (water-insoluble)	Collectors		Kerosene Fuel oil	Collects through selective wetting and adhesion of oil drops to coal particles
Surface active (water-soluble)	Frothers	Hydroxyl Nitrogenous	Aliphatic alcohols Pyridene containing tar oils	May also improve collecting ability of oily collectors
Emulsifiers	Promoters	Hydroxyl Carboxyl Nitrogenous	Polyethoxylated alcohols Fatty acids	Facilitate collector emulsification and its spread over coal surface
Inorganic	Modifiers	Water soluble salts	NaCl, CaCl ₂ , H ₂ SO ₄ , CaO	Promoters pH regulators Sulphide depressants
Protective colloids	Depressants	Hydroxyl Carboxyl	Polymers: starch, dextrin, carboxymethyl cellulose	Modifiers Coal depressants

Collectors are mixtures of hydrocarbon oils such as diesel fuel or of a paraffin type oil. The components of diesel are almost exclusively hydrocarbons of varying molecular weights, and there is a strong similarity with hydrocarbon groups on the coal surface. The mechanism of adsorption is now widely accepted as being hydrophobic bonding between the collector molecules and associated groups on the coal surface. This is a physical rather than a chemical bond. A simplistic diagram of the collector and attachment to a fine coal particle is presented in Figure 6.2 (Woollacott and Eric, 1994)

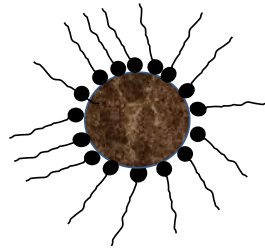


Figure 6.2: Collector and attachment to a particle (after Woollacott and Eric, 1994)

Bunt (1997) noted that conditioning of coals using oily collectors is also strongly dependent upon coal surface properties. From a thermodynamic point of view, the spreading of an oil film on a coal surface requires an energy input, termed the work of spreading. Oils spread more easily over high rank coals than over low rank coals. In addition, the structure and composition of the oil plays a role; aromatic oils containing surfactant impurities generally spread the most readily. However, enhanced oil spreading does not necessarily improve flotation recoveries.

Frother selection is also important: generally easily floated coal requires that only alcohol type frothers need to be utilized. As the natural floatability decreases, stronger frothers such as polyglycols and polyether alcohols may be more suitable.

Strong frothers cause a robust, slow draining froth to be formed. This certainly aids in the recovery of coarse particles, but can lead to more entrained ultrafine clay being held within the froth matrix due to poor drainage. This can be overcome with the use of froth wash water. Usage of strong frothers is recommended for oxidized coals. With weak frothers, the froth matrix is not held as rigidly as with strong frothers, and therefore selectivity is increased. However, there is a corresponding decrease in coarser particle recovery.

6.1.3 Dynamics and Cell Design

As stated earlier, the froth flotation process is a very complex one and is subject to the interaction to a large number of variables. One common theme from several textbooks (Woollacott and Eric, 1994), (Wills and Napier-Munn, 2006), (Kelly and Spottiswood, 1982) is that the flotation recovery is governed by air rates, impeller

speed, froth depths, the bubble surface flux area must be adequate to transport the desired mineral from the pulp, along with the floatability of the particles themselves.

To achieve this, the design of the flotation cell has undergone considerable evolution from the first inception such as the batteries of cells flowing into each other similar to that illustrated in Figure 6.3 (taken from Woollacott and Eric, 1994), to modern larger diameter tank cells which allows the required retention time to be achieved with fewer cells in the circuit. Wills and Napier-Munn (2006) also cite that a coal flotation circuit is typically less complex than that for other minerals and typically comprises only of a roughing stage, though a scavenger stage may be added.

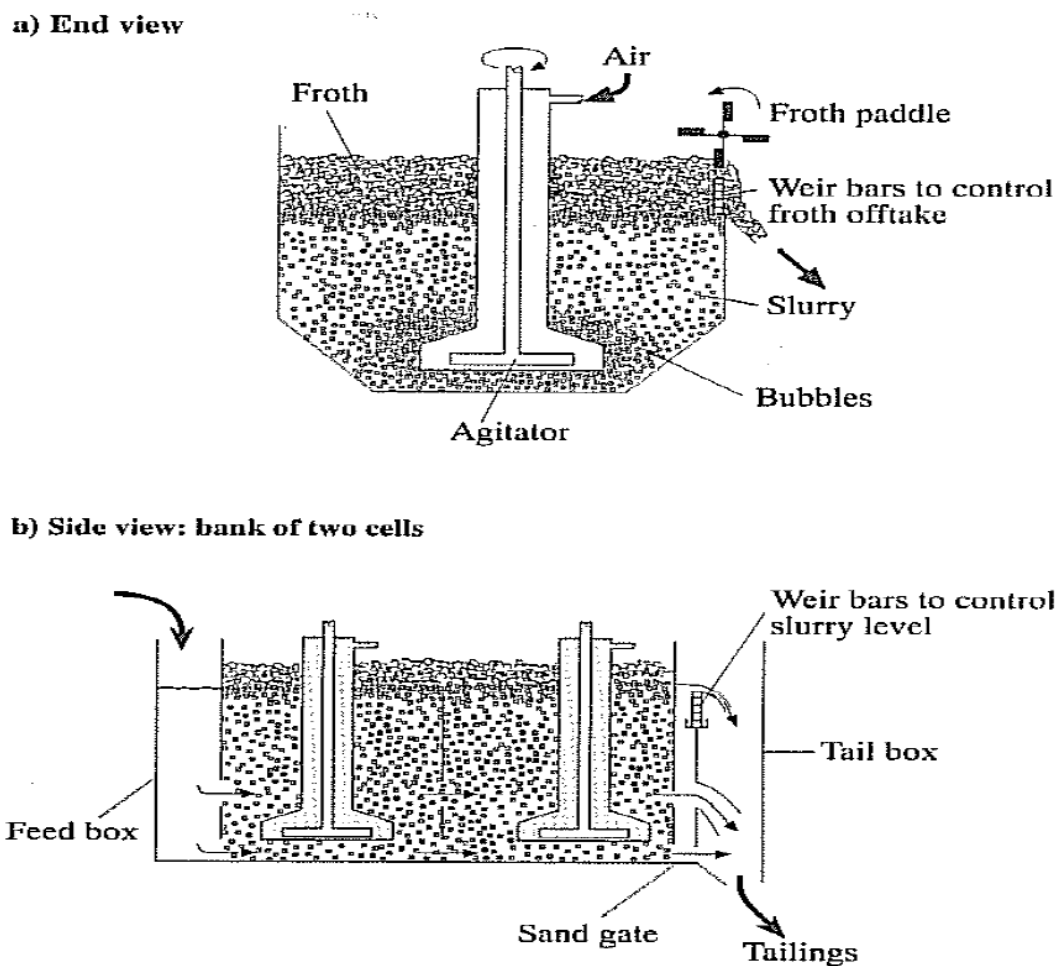
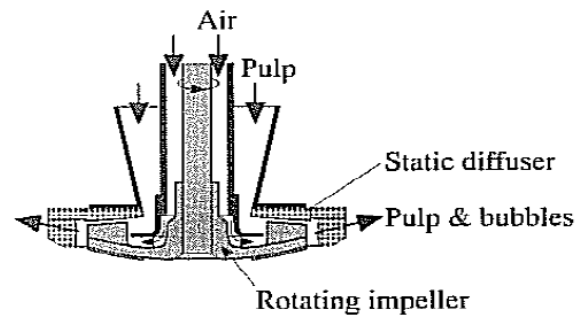
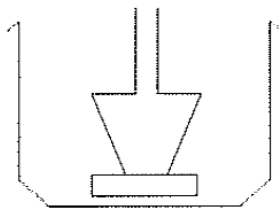


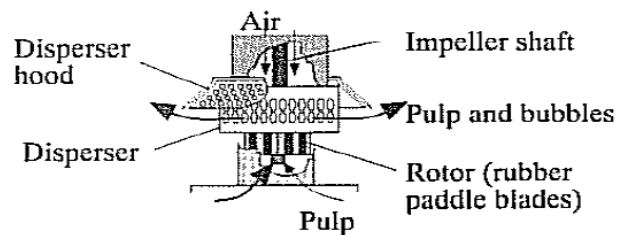
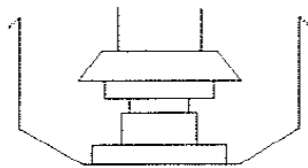
Figure 6.3: Early flotation cell design (Woollacott and Eric, 1994)

The internal design of the impellor and gas dispersion mechanisms have also evolved in an attempt to improve the probability of bubble-mineral contact. Three common designs are shown in Figure 6.4, (also taken from Woollacott and Eric, 1994), though no one design is generally superior to the others, but is rather other and site specific.

a) Denver



b) Wemco



c) Outokumpu

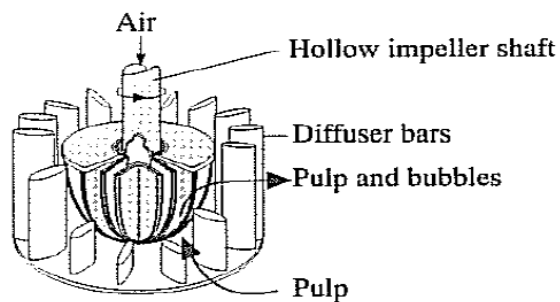
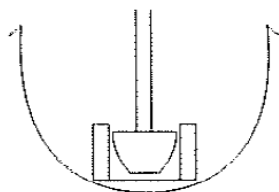


Figure 6.4: Different designs of flotation tank internals (Woollacott and Eric, 1994)

Column flotation technology, which would also include the Jameson cell, as depicted in Figure 6.5, has also been applied to coal flotation (Wills and Napier-Munn, 2006), albeit in few applications in South Africa.

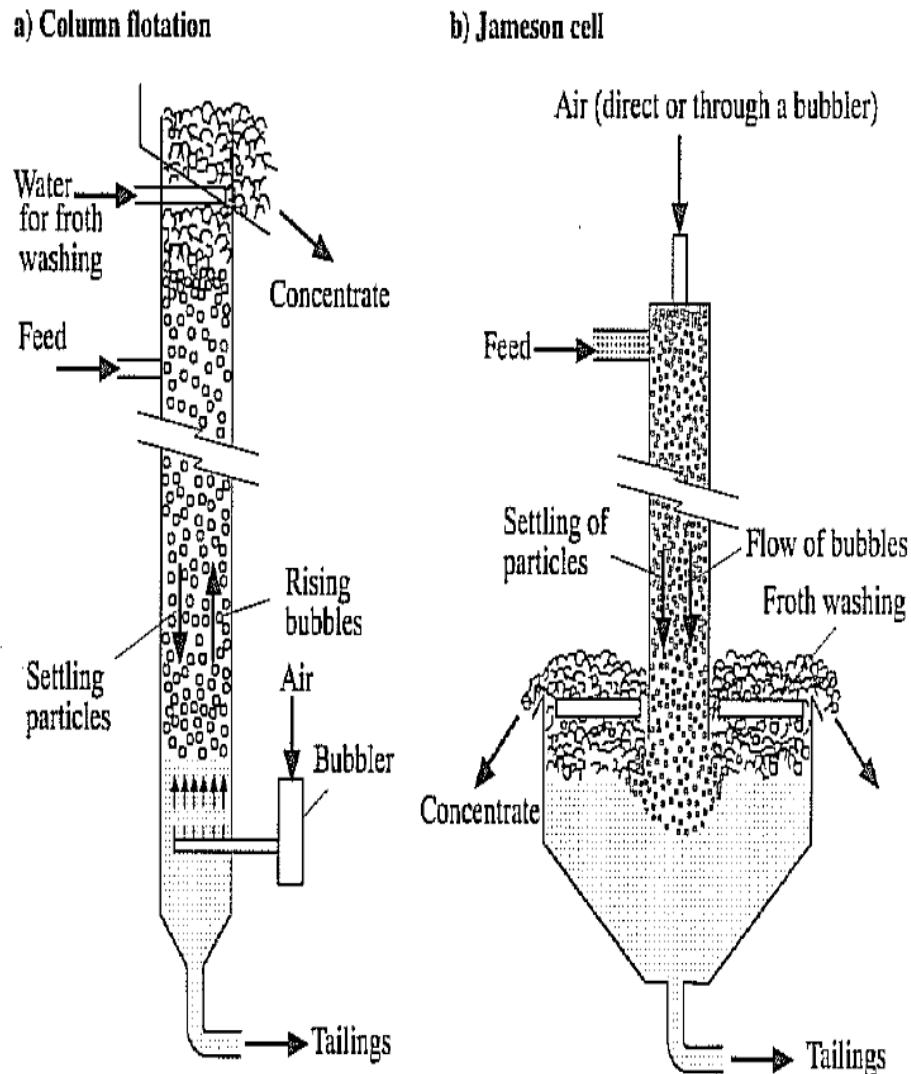


Figure 6.5: Comparison of a Column Cell and a Jameson Cell (Wills and Napier-Munn, 2006)

6.2 Flotation Experimental Test Procedures

6.2.1 Dump Borehole Samples

As described in section 3.2.2 above, each of the 1.5m long incremental core lengths were combined to form composite samples of 4.5m long. The composite samples were then air dried, weighed, mixed well and split so that three quarters of the sample was available for testing and the remaining quarter stored as a reference sample for any future work.

The analytical sample was then processed according to the procedure illustrated in Figure 6.6 (a recap of Figure 3.7 presented in section 3.2.2 above). This procedure was followed for the old dump, current dump, and the Visgat Dam material. However, it was noted that the central section of the old dump and also the Visgat Dam material did not contain any +0.5mm material and therefore only required milling prior to flotation.

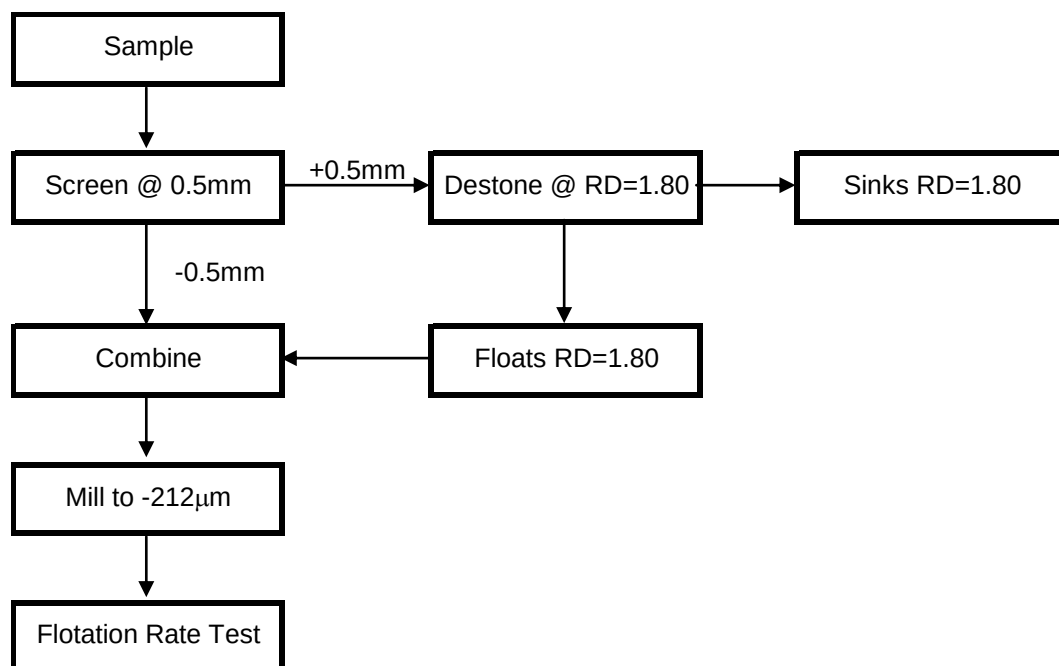


Figure 6.6: Borehole sample test procedure

Each 4.5m composite sample was screened at 0.5mm, the +0.5mm fraction was then destoned by subjecting it to a single density float and sinks step at a relative density of 1.80. The floats fraction was then combined with the -0.5mm fraction,

milled to $-212\mu\text{m}$ using a laboratory scale rod mill, which was 339mm long and 298mm in diameter, the rods used were 284mm long, and subjected to a flotation rate test. At each stage, a representative analytical sample was extracted, prepared and the ash content determined according to the relevant ISO standards.

The rate test was conducted by using 500g of the milled sample in a 1.8 litre Denver cell, the slurry was made up with tap water and was agitated for 2 minutes prior to the addition of 0.26mL (equivalent to approximately 0.7kg/t in the flotation cell) of the premixed reagent comprising of 85% paraffin/15% Betafroth BTN3 frother used to replicate the reagent suite used at Tshikondeni and conditioned for 2 minutes prior to floating. Concentrates were collected at timed intervals of 0-30 seconds, 30-60 seconds, 60-120 seconds and 120-240 seconds. Figure 6.7 shows a timed concentrate being collected for one of the tests. The impellor speed was set at 1200rpm throughout.

The concentrates and tailings were filtered, dried, prepared and analyzed for ash content.



Figure 6.7: Flotation test

6.2.2 Current Arising Discard – Monthly Composite Samples

The current arising discard was examined by means of composite samples made up from the daily production samples for the months of April 2008, May 2008, June 2008 and September 2008. These samples were processed as illustrated in Figure 6.8 (as also described in Figure 4.24 in section 4.2.2 above), which was similar to that for the boreholes, but with the inclusion of the crushing of +13mm material after noticing significant amounts of +13mm material were present in the samples. This would be expected in an open loop crushing circuit such as that used at Tshikondeni: i.e. the more friable coal would be expected to crush preferentially over any gangue minerals present in the run of mine material. These larger particles were also observed to be plate like as depicted in Figure 4.31 in section 4.3.3.

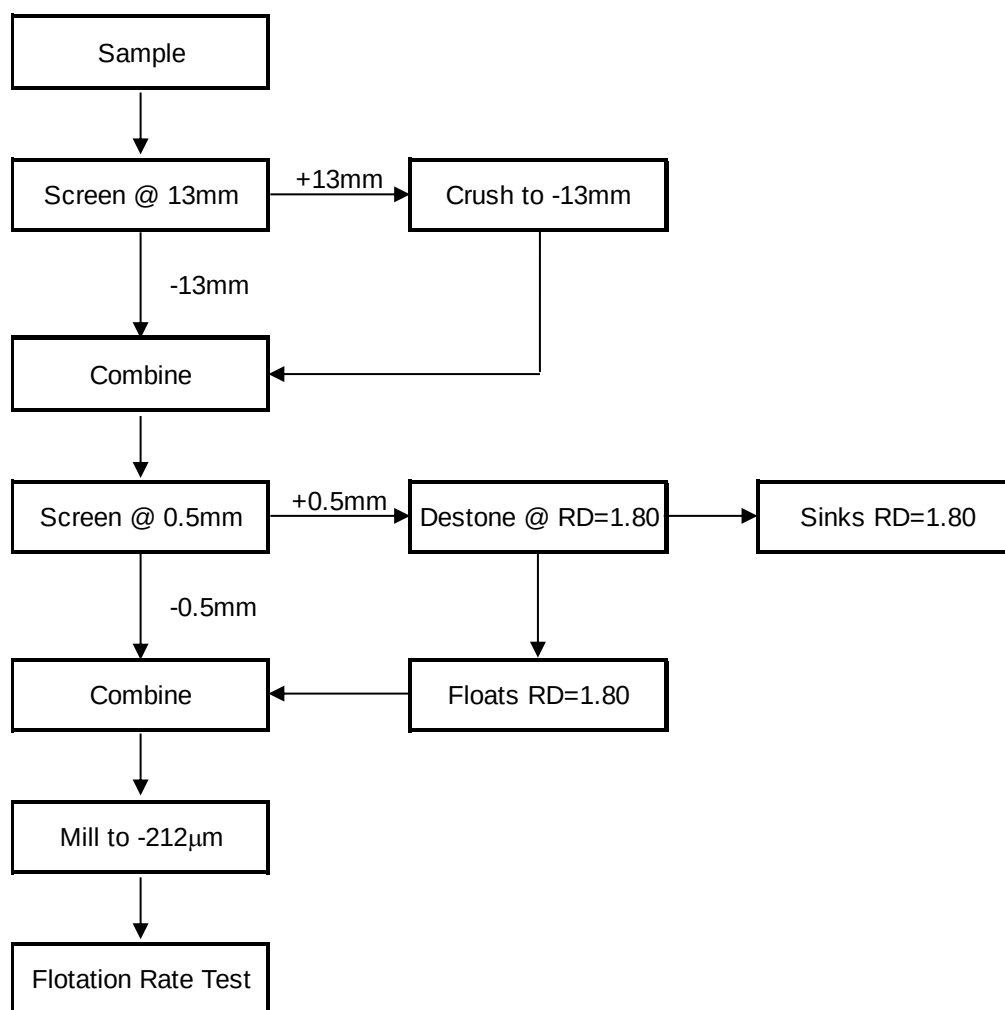


Figure 6.8: Arising discard test procedure

For the June 2008 and August 2008 samples, a comprehensive screening exercise was included in the test procedure. Each sample was screened at 20mm, 18mm, 12.5mm, 8mm, 3mm, 1.4mm, 0.6mm, 212 μ m, 125 μ m and 75 μ m. Thereafter the +20mm, -20+18mm and -18+12.5mm fractions were recombined and crushed to -13mm and the test procedure as per Figure 6.8 followed.

For comparative purposes, the -1.4mm material was also subjected to a flotation rate test to simulate what could be recovered under normal plant conditions, or the addition of additional plant flotation residence time in the form of a scavenger section. For the June and August samples, the arising -1.4mm material was milled to -212 μ m prior to the flotation rate test to examine potential for a regrind stage in the flotation circuit.

6.2.3 Old Discard Dump – 20 Tonne Bulk Sample

6.2.3.1 Rougher Rate Tests

From the work performed on the monthly composite samples and the bulk samples from each of the three shafts it was evident that very little value was contained in the coarser i.e. >4mm size fractions. This section of the work would therefore examine whether a saleable product could be produced from the -3mm material.

A sub-sample of the material was split from the bulk of the material, screened at 3mm and 0.5mm. The -0.5mm material was screened out since it would not be subjected to this process in a plant environment and would contaminate the density separation solutions used for laboratory scale dense medium separation. The -3+0.5mm fraction was destoned at a relative density of 1.75 and the -0.5mm material added to the floats 1.75 fraction to form the froth flotation feed. A 600g sub-sample of the froth flotation feed was milled for 6 minutes in a laboratory scale rod mill to -212 μ m, which was in turn divided into three lots of approximately 190g for use as the feed for the three flotation tests. The test procedure is depicted in Figure 6.9 and the reagent dosages for the three initial tests summarized in Table 6.2, paraffin was used as the collector and Betafroth FTN3 as the frother. Some changes

to the reagent dosages were made to see if an improvement in terms of selectivity i.e. lower concentrate ashes could be achieved. The tests were conducted at a solids content of 8% in tap water using a 1500mL Denver flotation cell. Timed concentrates were collected over 0-30 seconds, 30-60 seconds and 60-120 seconds which, along with the tailings were dried and the ash content determined. Water was added as required to maintain the level in the flotation cell.

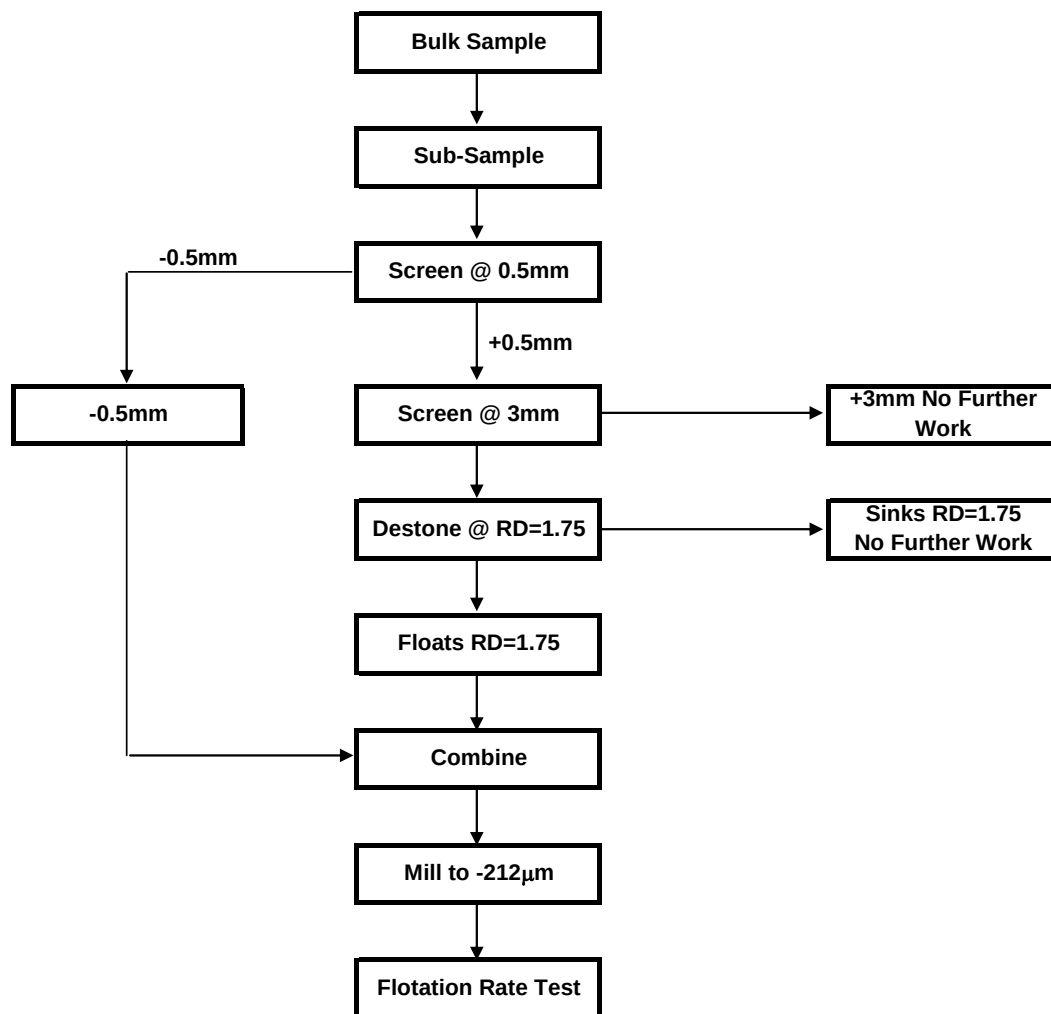


Figure 6.9: Test procedure

Table 6.2: Flotation reagent dosages

Test	Paraffin (g/t)	FTN3 (g/t)
Test 1	320	50
Test 2	160	25
Test 3	0	100

6.2.3.2 Cleaner Rate Test

Based on the similar results obtained from the above three tests, the remainders of the individual concentrates were combined together, milled for one minute in the rod mill without the addition of any water. The milled concentrate was then returned to the flotation cell and 150g/t paraffin added along with Cytec S-8860 a polyacrylamide based depressant recommended by the reagent supplier to further assist in improving the overall concentrate grade.

6.2.3.3 The Effect of Toluene and Tribromoethane on Flotation

Based on the initial results from the tests described above, and also the previous borehole and bulk samples, it was necessary to determine whether or not standard reagents to perform laboratory scale float and sinks procedures, namely tribromoethane and toluene, were able to influence flotation performance.

To investigate this, two sub-samples of approximately 600g of the -3mm discard, which had an ash content of 44%, were each subjected to a series of three tests:

- The first sub-sample was milled for 6 minutes in the rod mill and then split into three samples of approximately 190g for flotation testing.
- The second sub-sample was washed in a solution containing tribromoethane (TBE), toluene and white spirits. It was then filtered and washed again with white spirits to remove any residual TBE and toluene, dried at 65°C, milled for 6 minutes and divided into three samples of 190g each for flotation testing.

Tests were again conducted at a solids content of 35% in a 1500mL Denver flotation cell at the reagent dosages shown in Table 6.3, the same dosages for flotation reagents were used as was used for the untreated portion of material, so that the effect of the TBE and toluene additions could be investigated. . The flotation concentrate was collected as a bulk concentrate over 2 minutes for the first two tests, where it was observed at this time that the froth contained no solids. The third test used a flotation time of 1 minute to determine the yield and ash of the

concentrate before total flotation had occurred. In all three tests, water was added as required into the cell during the test to maintain the correct level.

Table 6.3: Flotation reagent dosages

Test	Paraffin (g/t)	FTN3 (g/t)	Flotation Time (min)
Test 1	0	25	2
Test 2	150	25	2
Test 3	150	25	1

6.2.3.4 Flotation Tests After Destoning By Dense Medium Separation

Having established that the use of TBE and toluene had an effect on the flotation performance of the discard, further test work was undertaken using magnetite and water to form a dense medium suspension in a bucket. Before the coal was added to the slurry, the stirring rate was decreased significantly in order to prevent the coal from being subjected to excessive centrifugal forces, which could influence the rate of density related separation. It should be noted that the densities of the magnetite suspension could have changed during the upgrading process due to the decreased stirring rate. Furthermore, the material was removed subjectively, since the separation level was not controlled and the coal that floated was removed manually. The material that floated was used as the feed for froth flotation tests. The general arrangement of the apparatus used is depicted in Figure 6.10. Note that differences in separation density for the tests were experienced and accordingly attributed to the technique.

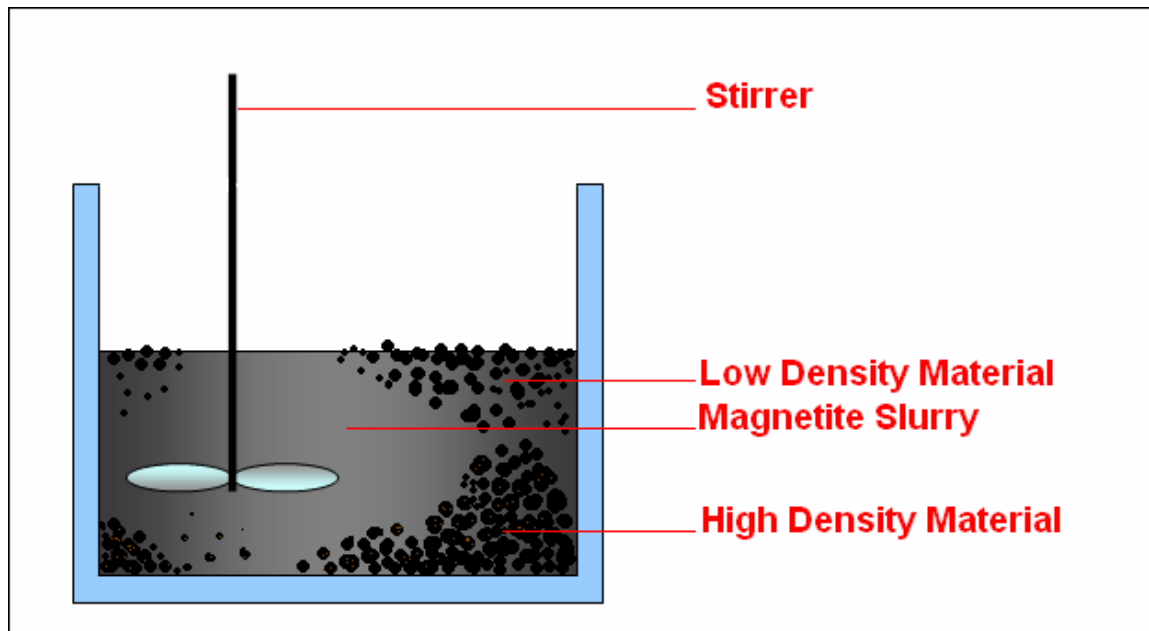


Figure 6.10: Dense medium separation of coal using a magnetite suspension

The representative sub-samples of the discard were spilt from the bulk material and were subjected to four different dense medium separation scenarios to generate a feed sample for froth flotation testwork:

1. The discard was screened at 0.5mm and 3mm, the $-3+0.5\text{mm}$ material was upgraded at a relative density of 1.75, and the -0.5mm material added back to the floats 1.75 fraction to produce a flotation feed with an ash content of 24.2%
2. The discard was screened at 0.5mm and 6mm, the $-6+0.5\text{mm}$ material was upgraded at a relative density of 1.68, and the -0.5mm material added back to the floats 1.68 fraction produce a flotation feed with an ash content of 29.0%
3. The discard was screened at 0.5mm, and the $+0.5\text{mm}$ discard was upgraded at a relative density of 1.72, and the -0.5mm material added back to the floats 1.72 fraction produce a flotation feed with an ash content of 29.5%
4. The discard was screened at 0.5mm, and the $+0.5\text{mm}$ discard was upgraded at a relative density of 1.65, and the -0.5mm material added back to the floats 1.65 fraction to produce a flotation feed with an ash content of 24.4%

As in previous tests, the flotation feed sample was milled to -212µm and split into batches of approximately 190g prior to froth flotation in tap water using a 1.8L Denver flotation cell. Paraffin and CRX 11 were used as collectors, Betafroth FTN4 frother and Cytec S-8860 GL was used as a depressant. The use of Betachem CRX11 and Betafroth FTN4 was at the recommendation of the supplier with a view to improved separation selectivity. The reagent conditions are summarized in Table 6.4.

Table 6.4: Matrix of tests and test conditions

Sample	Reagent Dosage (g/t)				Test Type
	PARAFFIN	CRX 11	FTN 4	Cytec S-8860 GL	
-3mm Discard F1.75	0	0	150	0	Rougher test
	150	0	150	0	Rougher test
	150	0	150	300	Rougher test
-3mm Discard F1.68	100	0	50	0	Rougher test
	100	0	50	100	Rougher test
	150	0	50	100	Rougher test
	0	200	100	0	Cleaner test
Whole Discard F1.72	100	0	100	0	Rougher test
	0	200	0	0	Rougher test
	0	400	100	0	Rougher test
	0	200	200	0	Rougher test
Whole Discard F1.72	0	200	400	0	Rougher test

6.2.3.5 Flotation Test in the -0.5mm Size Fraction

The section of the work was undertaken to consider the effect of froth flotation on the fines fraction of the discard produced by Tshikondeni. This size fraction is not easily processed in a standard coal preparation plant and, moreover, under standard laboratory float and sinks procedures it is difficult to recover, takes a long time to settle out, and more importantly contaminates the separation medium.

The conditions for the various tests are summarized in Table 6.5 below.

Table 6.5: Matrix of tests and test conditions on -0.5mm discard

Sample	Reagent Dosage (g/t)			Test Type
	PARAFFIN	CRX 11	FTN 4	
-0.5mm No Milling	200	200	400	Bulk Rougher Test
-0.5mm Milled to -212µm	200	400	400	Bulk Rougher Test
-0.5mm No Milling	0	100	100	Rougher Rate Test
-0.5mm No Milling	0	200	100	Rougher Rate Test
-0.5mm No Milling	0	100	100	Bulk Rougher Tests x7
-0.5mm No Milling	100	0	100	Bulk Rougher Tests x4
Rougher Concentrate Milled for 1 minute	100	0	100	Bulk Cleaner Test
Rougher Concentrate Milled for 2 minute	100	0	100	Bulk Cleaner Test
Rougher Concentrate Not Milled	50	0	50	Bulk Cleaner Test
Rougher Concentrate Not Milled	100	0	100	Bulk Cleaner Test

The first two tests were to compare the effect of milling the -0.5mm material down to -212µm and a bulk rougher test as described previously was performed. This was followed by a pair of rougher rate tests where the effect of reagent was varied: i.e. the reagents were reduced based on the high ashes achieved from the first pair of tests. Thereafter a series of seven bulk rougher tests were undertaken to determine the repeatability of the test.

The second set of tests were performed using paraffin instead of CRX11 as the collector and a further four bulk tests were performed to produce material for the bulk cleaner tests. Two cleaner tests were undertaken at two different reagent dosages, as well as two tests where the rougher concentrates were milled for one minute and two minutes respectively to see if milling produced a better concentrate due to creation of new surfaces by grinding slightly finer than -0.5mm.

6.3 Flotation Test Results

6.3.1 Dump Borehole Samples

6.3.1.1 Old Dump Results

Table 6.6 shows the average, maximum and minimum ashes and yields along with the standard deviations for the various steps taken in processing the samples. The use of a destoning process at a relative density of 1.80 was noted to give an average mass reduction of approximately 42%, whilst improving the ash from approximately 50% in situ to the order of 30%. The -0.5mm material accounted for approximately 24% of the mass in the dump at an ash content of 39%, and when combined with the floats 1.80 product generated an average flotation feed representing approximately 58% of the mass at an ash content of 34%.

Table 6.6: Summary of the basic results after processing the dump samples

	Average		Standard Deviation		Maximum		Minimum	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
Insitu Discard	100.0	50.3	-	2.8	100.0	61.6	100.0	40.4
+0.5mm Material	76.4	53.4	5.0	3.0	96.3	65.0	57.8	39.8
+0.5mm Floats RD1.80	34.3	29.8	4.4	1.9	51.5	38.6	18.5	27.6
+0.5mm Sinks RD1.80	41.9	72.9	5.5	1.1	61.5	78.0	19.0	70.2
-0.5mm Material	23.6	38.6	5.0	5.1	42.2	48.1	3.7	23.1
Flotation Feed	57.9	34.1	5.6	2.1	81.0	43.5	38.5	29.1
Flotation Product	25.5	19.3	5.2	1.8	38.3	29.9	11.0	16.6
Flotation Tailings	32.0	45.9	5.9	5.0	53.8	62.4	8.7	19.4
Final Tailing	74.2	61.1	5.3	2.9	89.0	69.2	61.7	45.5

The flotation rate test results for yield versus time and ash versus time are graphically presented in Figure 6.11 and Figure 6.12 below. For comparative purposes, both figures also contain the response for a typical flotation plant feed and flotation plant discard from the August 2008 plant historical data.

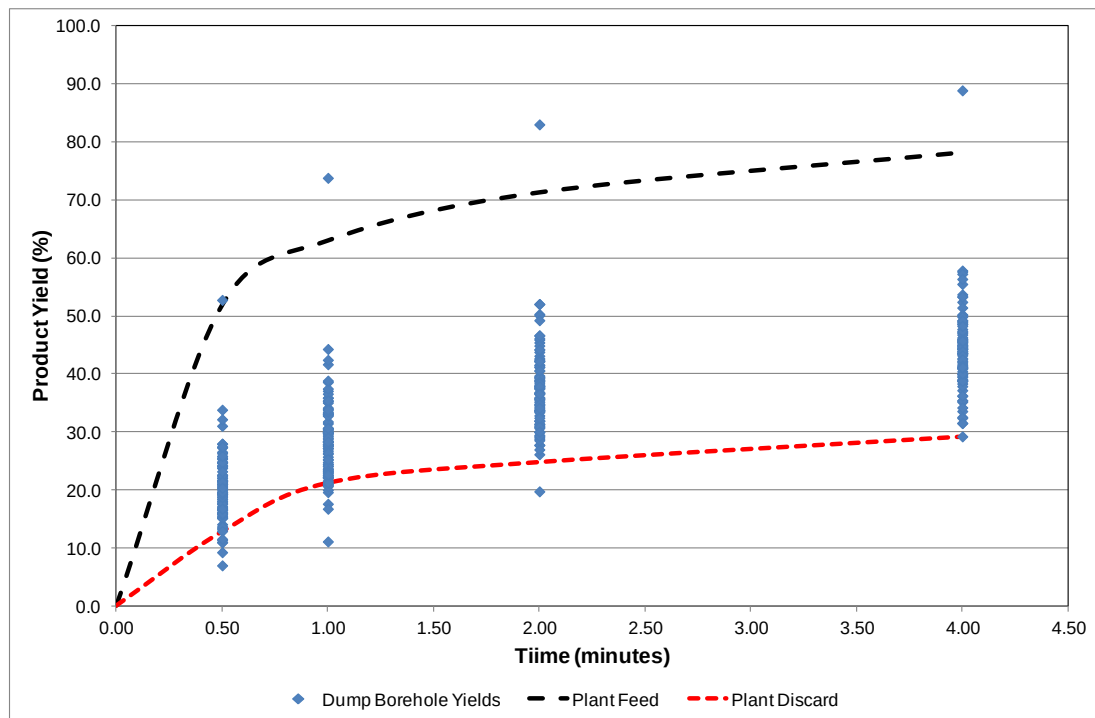


Figure 6.11: Flotation rate concentrate yields versus time for the all the samples collected

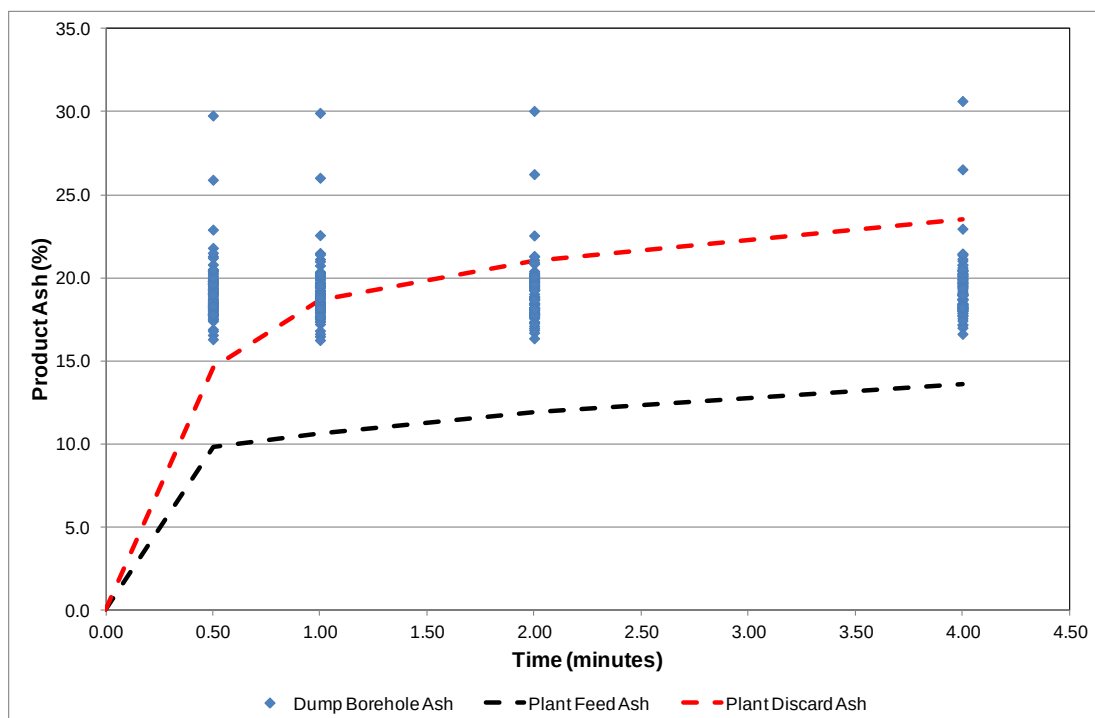


Figure 6.12: Flotation rate concentrate ash contents versus time for the all the samples collected

An extremely wide range of yields were noted which indicated that the standard flotation tests were not selective and as is illustrated by the ash results, the quality target of 14% was not achieved. The majority of the yields noted for the borehole samples were, as would be expected, within the envelope demarcated by the flotation plant feed and flotation discard yield, though tended to be generally closer to that of the plant discard. The ash results were noted to be much higher than that of the typical flotation discard, and none were within the envelope at a 30 seconds of flotation due to a combination of high mass pulls and poor flotation selectivity.

Yield release curves from the four sections produced by Borehole 1, which showed a typical flotation rate response, are illustrated in Figure 6.13 overleaf to further illustrate the flotation response of the borehole samples. The first 30 seconds of flotation was observed to generate approximately half of the mass yield at around 20%, with final yields of between 32% and 42% being achieved. The ash was fairly constant across all four timed concentrates and indicated that little selectivity was achieved. The work performed by Barnwal et al (2000) indicated that at low collector dosages the vitinite component floated faster than the inertinite component, but the ashes were noted to be higher than expected: this could possibly be attributed to ash (mineral matter) being pulled up by a hydraulic mechanism since bench testing is done at relatively low flotation cell volumes with a high specific energy input from the impeller. They also concluded that at below 0.5mm that the macerals should be well liberated, since these tests were conducted at -0.212mm, it would be expected that good maceral liberation had been achieved though it was not confirmed through petrographic examination. The shape of the yield curve suggests that a fast floating (possibly vitrinite and mineral matter) and slow floating (possibly inertinite and mineral matter) components were present in the destoned discards. The flat ash curve profile suggests that a lot of mineral matter was recovered initially, and that extensive work would be required to optimize the flotation process to achieve acceptable yields and ash grades. Additionally, several samples along with the plant feed and plant discards samples should have been subjected to a release analysis to obtain a better understanding of the flotation behaviour and eliminate the effects of entrainment.

Interpolating the yield and ash data to give a 14% ash product, suggested that a yield of approximately 10% would be expected, which is obviously too low to justify the capital expenditure required for the installation of a full scale plant to screen, destone, mill and float the discard dump. From the data observed, it is likely that a cleaner flotation stage would be required to upgrade the rougher concentrate and would add to both the costs from a capital and operating costs perspective.

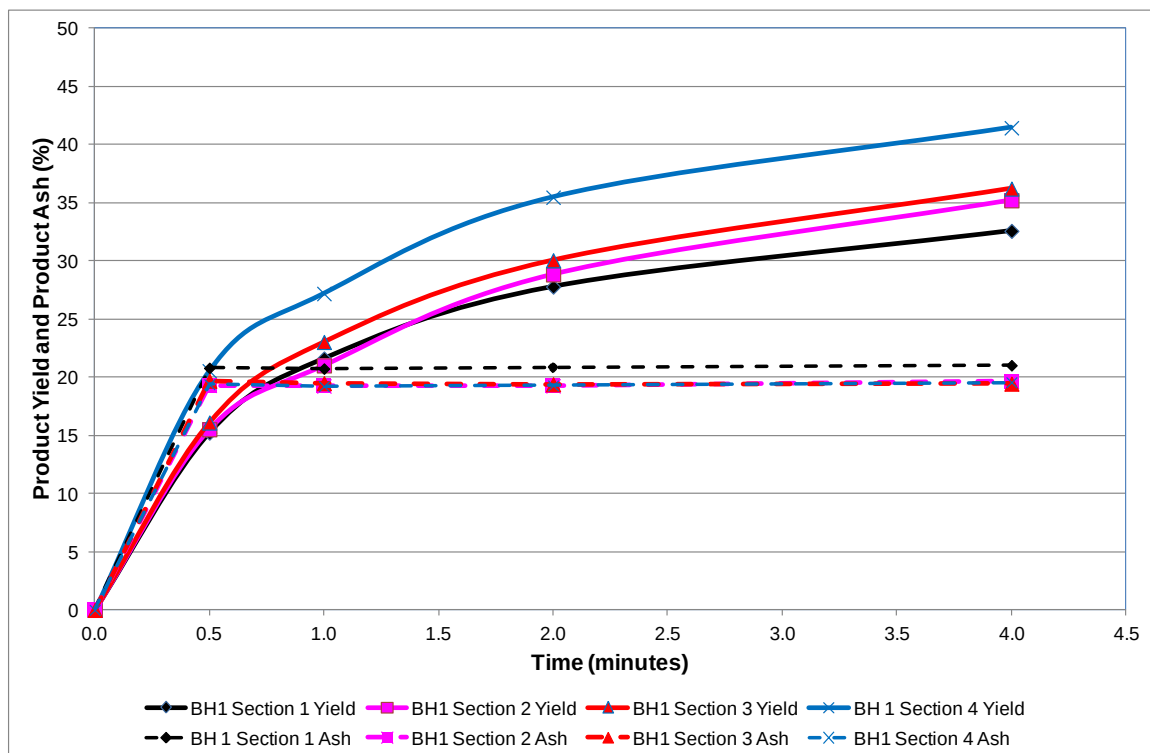


Figure 6.13: Flotation rate test for Borehole 1 showing response of all the sections

The overall shape of the yield curves depicted in Figure 6.13, suggested that the samples had a very large slow floating component, and that high recoveries even after long flotation residence times would be unlikely; to quantify this, two flotation models were considered for BH1. The first being the Klimpel model (Klimpel, 1980) model shown as Equation 6.1 which uses two parameters to describe the flotation rate kinetics and is of the form:

$$Recovery = R(1 - \frac{1}{kt}(1 - e^{-kt})) \dots \dots \dots \text{Equation 6.1}$$

where R is the recovery (yield) at infinite or maximum possible recovery (parameter 1), k is the flotation rate constant (1/min, and parameter 2), whilst t is the flotation time.

The second model considered was developed by Kelsall (1961) and accounts for fast and slow floating components and uses 3 parameters and is of the form:

$$\text{Recovery} = (1 - Q_s)(1 - e^{-k_f t}) + Q_s(1 - e^{-k_s t}) \dots \text{Equation 6.2}$$

where Q_s is the slow floating component (parameter 1), k_f is the fast floating rate constant (1/min, parameter 2), k_s is the slow floating rate constant (1/min, third parameter) and t is flotation time.

The models were also fitted to the typical flotation feed and discard samples and the results summarized in Table 6.7. The Klimpel model fits suggested that the overall recovery (yield) the sections of BH1 would be in the order of 35% to 44%, which was slightly higher than that observed for the plant flotation discard at 33%. Rate constants for all BH1 sections and the flotation discard were similar. The plant flotation feed was noted to be much higher, as expected, and is in line with yields obtained from earlier sections of the chapter.

The Kelsall model fits showed that the proportion of slow floating material within the borehole samples was very high at between 74% to 80%, similar to that of the arising flotation discard material (and approximately half of that for the flotation feed), suggesting a low probability of obtaining a product of grade suitable for the production of coke. The borehole rate constants were noted to be marginally higher than the plants flotation discard.

Table 6.7: Flotation model rate data

Sample	Klimpel Model		Kelsall Model		
	R	k	Q_s	k_s	k_f
BH1 Section 1	35.584	2.409	0.769	0.033	1.831
BH1 Section 2	38.051	2.133	0.804	0.055	2.201
BH1 Section 3	39.702	2.208	0.763	0.045	1.848
BH1 Section 4	43.984	2.655	0.735	0.058	2.308
Plant Discard	33.012	2.294	0.746	0.013	1.387
Plant Feed	81.429	4.804	0.403	0.157	3.374

This behaviour was also observed in other tests for subsequent sections of the work and the effect of organic liquids used for the heavy liquid separation was considered to be a major contributing factor in the outcomes observed. The effect of organic heavy liquid separation on the froth flotation tests is discussed in more detail in section 6.3.3.3 in this chapter.

As previously indicated, Boreholes 8 and 9 were sourced from the central fines section of the dump and three sections from each were drilled by the auger. No material above 0.5mm was found in the samples and therefore each section only needed to be subjected to froth flotation after milling to $-212\mu\text{m}$. Given that the flotation section of the plant performs well on the finer sized particles, as shown in section 2.4 in Chapter 2, it would be expected that very little room for recovery of a saleable product from this section of the dump was likely. Moreover, the length of time the material had spent in the dump would also have affected the particle surfaces, and some degree of oxidation/weathering would have been expected, though the effect of milling should have improved the floatability by creation of fresh surfaces. This concept was borne out by the flotation rate tests and the results depicted in Figure 6.14 and Figure 6.15 for BH8 and BH9 respectively.

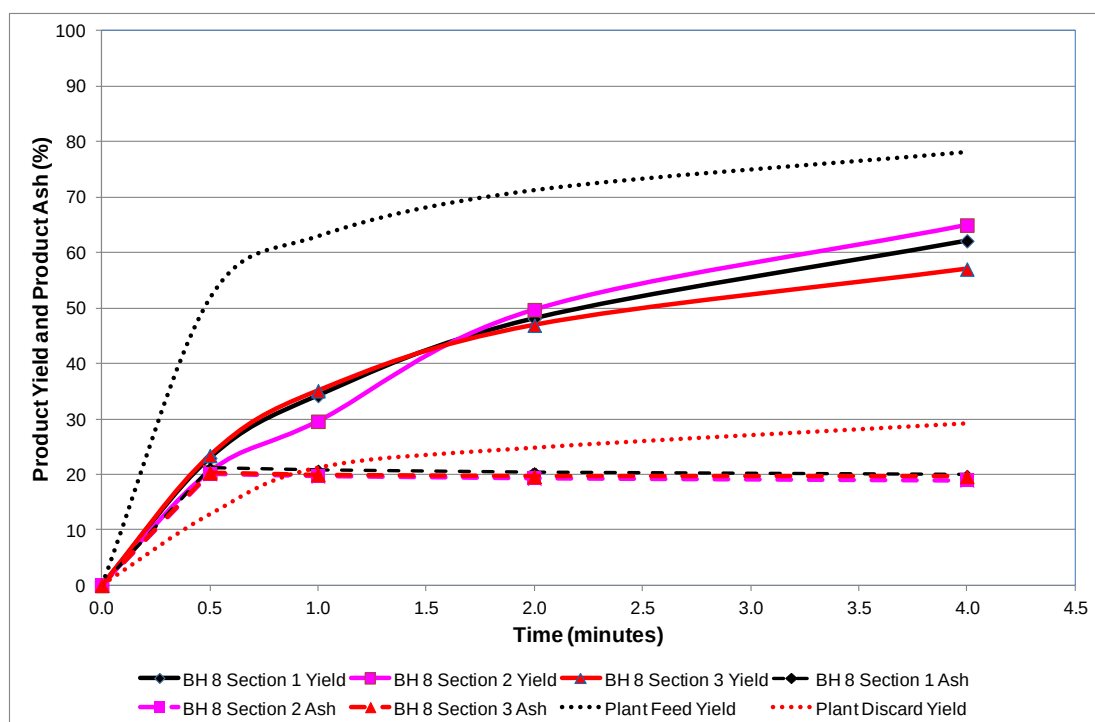


Figure 6.14: Flotation rate test Borehole 3 showing response of all borehole sections

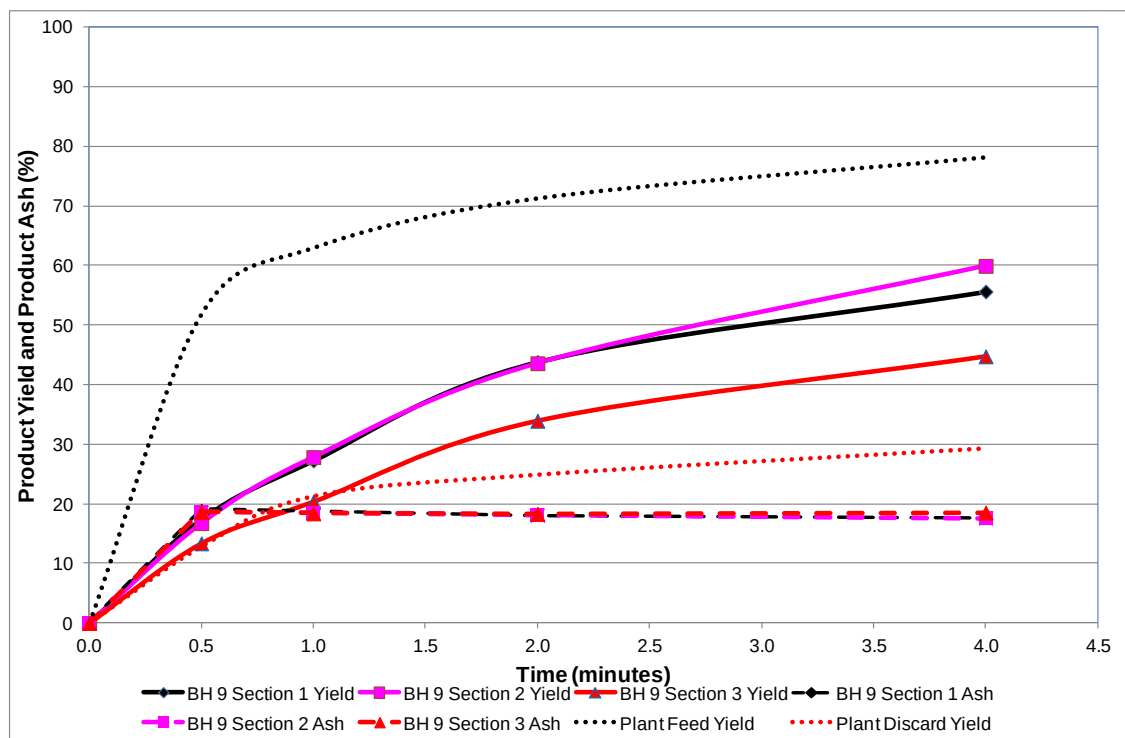


Figure 6.15: Flotation rate test Borehole 1 showing response of all borehole sections

The ashes produced from all of the timed concentrates were similar, and high at around 18-20%. The initial 30 second concentrate was observed to be the (marginally) highest ash for all six tests and are attributed to the high mass pulls experienced in the initial stages of the test.

Both samples also showed that an ash content of 14% would be produced at approximately 15% yield. Final product yields were higher than for the coarse material from the dump and would be expected since it has been previously determined that the higher ash waste material resides in the coarser size fractions.

The flotation rate model fits presented in Table 6.8 indicated that ultimate recoveries, which varied from 56% to 79%, were in line with that of the flotation feed, though the rate constants were much lower. Despite the Kelsall model indicating large proportions of slow floating species within the samples, the rate constant for the slow floating species was higher than that for BH1 and about an order of magnitude greater than that of the discard, being similar to that of the flotation plant feed. This is most likely attributable to the larger particles, possibly in the 300-500 μ m size

range being floated. These particles are considered to be middlings type particles as suggested by Klassen and Mokrousov (1963) and needed additional residence time in the plant's flotation circuit to be recovered, or were recovered into the froth and subsequently lost from the froth by turbulence within the cells. This avenue needs to be further explored in any subsequent work on the dump.

Table 6.8: Flotation model rate data

Sample	Klimpel Model		Kelsall Model		
	R	k	Q_s	k_s	k_f
BH8 Section 1	71.057	1.595	0.716	0.161	1.876
BH8 Section 2	78.396	1.160	0.750	0.191	1.419
BH8 Section 3	64.211	1.924	0.626	0.095	1.581
BH9 Section 1	71.041	1.125	0.614	0.088	0.923
BH9 Section 2	78.850	0.973	0.728	0.152	1.073
BH9 Section 3	56.169	1.064	0.791	0.093	1.191
Plant Discard	33.012	2.294	0.746	0.013	1.387
Plant Feed	81.429	4.804	0.403	0.157	3.374

Improvements in terms of both quality and yields could potentially be made by optimization of the reagent suite, or addition of a cleaner stage though it is unlikely to make a significant improvement in economic terms.

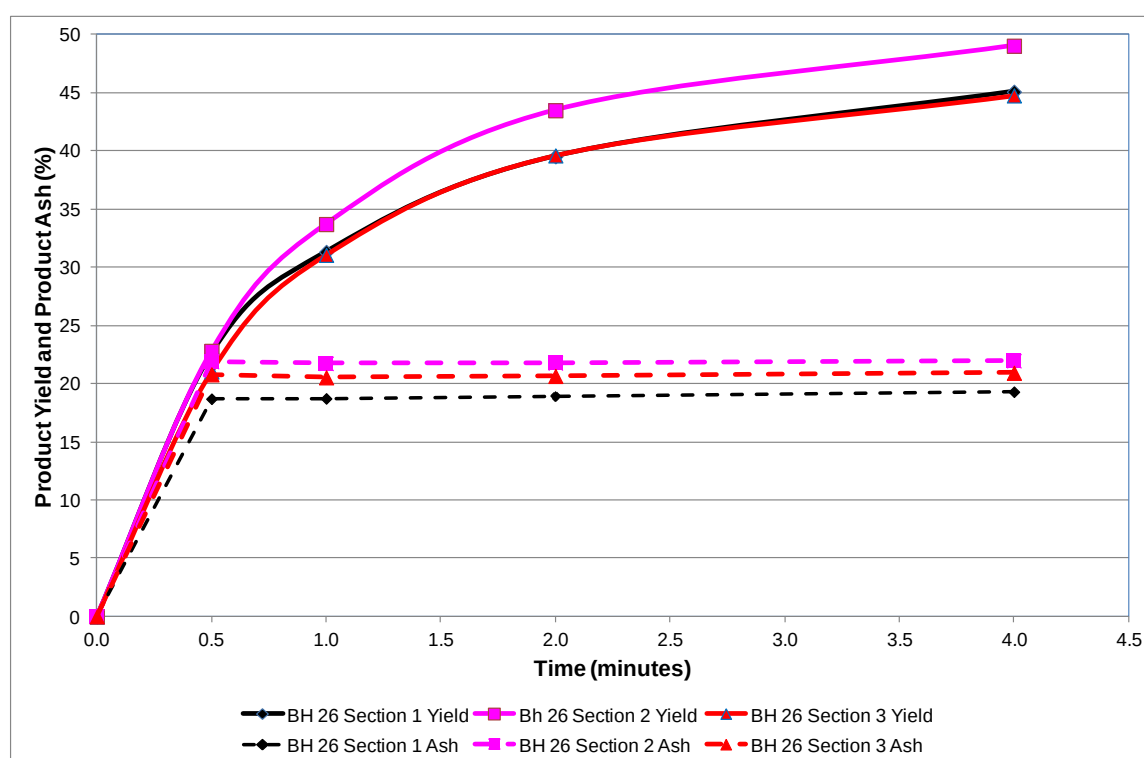
6.3.1.2 Current Dump Results

Table 6.9 shows the average, maximum and minimum ashes and yields along with the standard deviations for the various steps taken in processing the samples. These samples were observed to contain less -0.5mm than the old dump (18% cf. 24%), destoning rejected some 54% of the in situ mass at an acceptable 73% ash. The destoned material represented approximately 28% of the in situ mass with an ash content of 30%, and when combined with the -0.5mm material generated an average flotation feed of 46% of the initial mass at an ash of 35%.

The results showed similar behaviour to the coarse discard on the old dump and that the proposed screen, destone, mill and float process route would also not be suitable as illustrated in Figure 6.16 where the flotation rate test data show that a specification grade of 14% ash would give a yield in the order of 15%.

Table 6.9: Summary of the basic results after processing the dump samples

	Average		Standard Deviation		Maximum		Minimum	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
Insitu Discard	100.0	55.8	-	1.7	100.0	58.0	100.0	52.4
+0.5mm Material	81.7	58.7	1.2	1.6	83.0	60.8	79.4	55.3
+0.5mm Floats RD1.80	28.0	30.4	2.5	0.9	33.0	31.5	25.0	28.8
+0.5mm Sinks RD1.80	53.7	73.4	2.9	0.8	57.8	74.7	48.0	72.2
-0.5mm Material	18.3	42.9	1.2	2.1	20.6	47.0	17.0	40.2
Flotation Feed	46.3	35.4	3.0	1.1	52.1	36.8	42.2	33.2
Flotation Product	18.8	20.3	3.0	1.6	22.7	22.2	13.1	17.3
Flotation Tailings	27.5	46.1	5.1	3.6	39.0	49.4	22.5	37.4
Final Tailing	81.2	64.1	3.0	3.0	86.9	66.8	77.3	57.2

**Figure 6.16: Flotation rate test Borehole 26 showing response of all four borehole sections**

The flotation rate model parameters were determined for BH26 and summarized in Table 6.10. The model fits showed better overall kinetics to those of BH1 from the coarse discard on the old dump. The Klimpel model recoveries were determined to be between 49-54% compared to 33-44% for BH1, though the rate constants were similar for both samples at between 2.4-2.7. The proportion of slow floating material was also determined to be lower at 59% to 67% compared to the 74-80% returned

for BH1. The rate constants for the Kelsall model were similar for both BH26 and BH1.

Table 6.10: Flotation model rate data

Sample	Klimpel Model		Kelsall Model		
	R	k	Q_s	k_s	k_f
BH26 Section 1	48.755	2.727	0.665	0.049	1.943
BH26 Section 2	54.732	2.380	0.589	0.037	1.496
BH26 Section 3	49.897	2.426	0.628	0.033	1.539
Plant Discard	33.012	2.294	0.746	0.013	1.387
Plant Feed	81.429	4.804	0.403	0.157	3.374

6.3.1.3 Visgat Dam Results

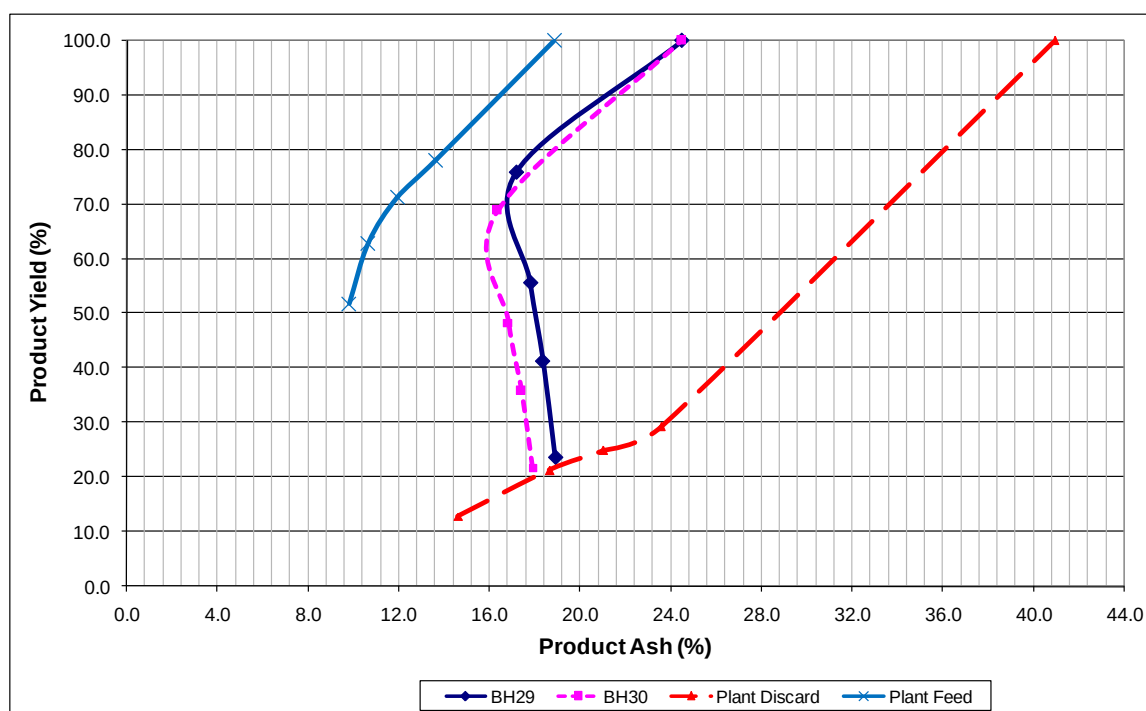
These two samples were found to be composed entirely of -0.5mm, which was to be expected as the material came from solids suspended in a return water dam, likewise the relatively good feed ash contents at 24.5%. Despite the relatively good quality of the feed ashes, it should be borne in mind that the material in this dam is the result of many years of build-up and had been left from the early 2000s, and would be expected to have some degree of surface oxidation. Hence, these samples did not require any destoning and were floated without any pre-concentration after milling to -212 μ m, which was intended to produce fresh particle surfaces and mitigate to some extent any surface tarnishing. The flotation rate test results are presented in Table 6.11 and Table 6.12 for BH29 and BH30 respectively, with the yield-ash relationship illustrated in Figure 6.17, which also shows the relationships obtained from similar tests performed on the monthly flotation feed and discard samples from August 2008, though this had a higher ash content at 40.9%. Of interest is that the ashes for the timed concentrates reduce as the test progresses and was attributed to the large mass pulls experienced, particularly in the first minute of flotation. Neither of the two tests was able to generate a product at the desired 14% ash specification, though this would probably be achievable by either optimizing the reagent suite or by means of a rougher-cleaner operation.

Table 6.11: Rate test data for BH29

Stream	Time (min)	Yield (%)		Ash (%)	
		Fractional (%)	Cumulative (%)	Fractional (%)	Cumulative (%)
Conc 1	0.50	23.7	23.7	18.9	18.9
Conc 2	1.00	17.7	41.4	17.6	18.3
Conc 3	2.00	14.3	55.7	16.2	17.8
Conc 4	4.00	20.2	75.9	15.4	17.2
Tails	-	24.1	-	47.5	-
Feed	-	100.0	100.0	24.5	24.5

Table 6.12: Rate test data for BH30

Stream	Time (min)	Yield (%)		Ash (%)	
		Fractional (%)	Cumulative (%)	Fractional (%)	Cumulative (%)
Conc 1	0.50	21.8	21.8	17.9	17.9
Conc 2	1.00	14.0	35.8	16.6	17.4
Conc 3	2.00	12.6	48.4	15.2	16.8
Conc 4	4.00	20.6	68.9	15.2	16.3
Tails	-	31.1	-	42.5	-
Feed	-	100.0	100.0	24.5	24.5

**Figure 6.17: Yield-ash relationship**

Flotation rate data, following the application of the Klimpel and Kelsall flotation rate models, is summarized in Table 6.13 with the yield-flotation time relationships shown

in Figure 6.18. The August 2008 composite flotation plant feed and discard sample are also included for comparison.

Table 6.13: Flotation model rate data

Sample	Klimpel Model		Kelsall Model		
	R	k	Q_s	k_s	k_f
BH29	90.330	1.304	0.634	0.236	1.259
BH30	81.364	1.245	0.773	0.221	1.899
Plant Discard	33.012	2.294	0.746	0.013	1.387
Plant Feed	81.429	4.804	0.403	0.157	3.374

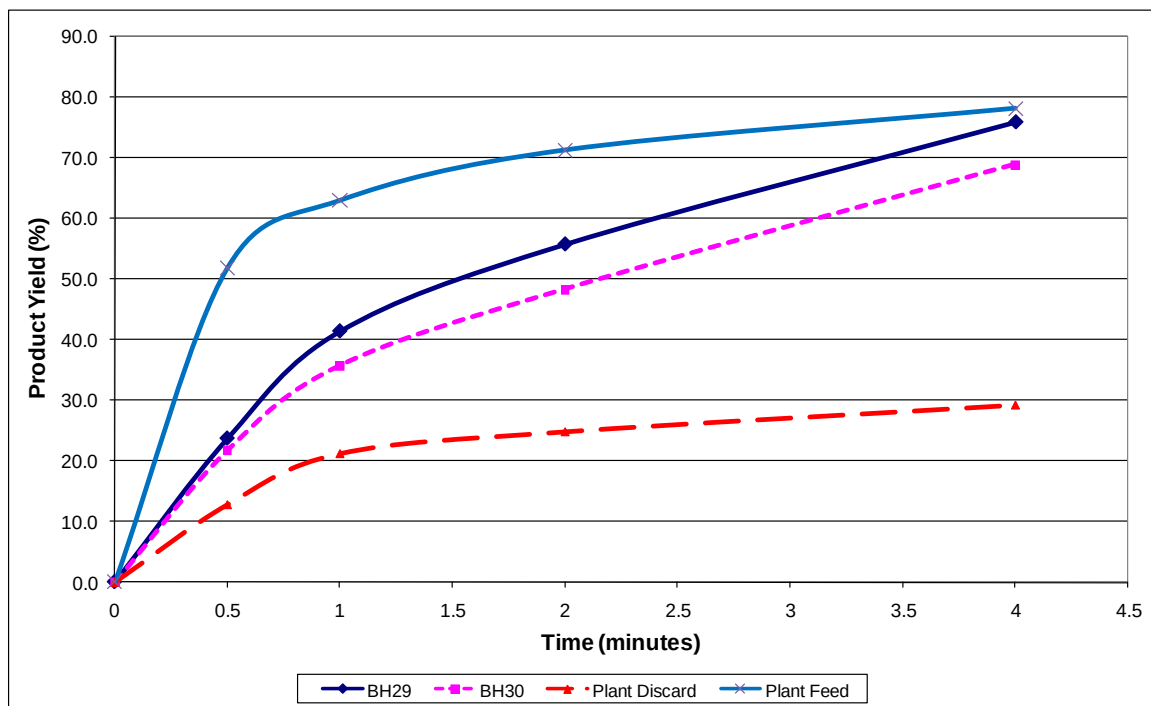


Figure 6.18: Yield vs. flotation time relationship

The Klimpel model indicated that recoveries in excess of 80% are achievable from the material from the Visgat samples, though the rate constant was noted to be slow; a typical feed sample would be in the order of 4.8. Probably more importantly, the Kelsall model, which takes into account both fast and slow floating species, indicated that both borehole and the plant discard samples contained significant quantities of slow floating species at between 63-77%, whilst the feed contained only 40% of slow floating material, and it is noticed that most of the flotation occurred within the first minute of flotation at an ash content of approximately 10%. Given that the plant treats -1.4mm material in the flotation section, and plant testwork has shown that the

losses from the flotation section are typically in the coarser (-1.4+0.5mm) size range, which would account for the slow rate data for the discard material.

The Visgat material could potentially be upgraded to a saleable product level, though reagent optimization would be required and a large increase in yield would probably not be possible. Upon mine closure, the area would need to be rehabilitated and a potential alternative would be to consider “mining” this material and slowly adding into the plant run-of-mine over a period of several months and allowing the plant to recover any saleable product directly.

Another alternative would be to consider the sale/supply of the material to the two local brickworks currently purchasing small volumes of the Tshikondeni arising discards in their clay brick production as a filler material in their processes.

6.3.2 Current Arising Discard – Monthly Composite Samples

The detailed results for the current arising discard samples taken over the period April 2008 to August 2008 are included in Appendix 5, whilst Table 6.14 gives a summary of the results achieved.

Table 6.14: Summary of results

Stream	April 2008		May 2008		June 2008		August 2008	
	Mass (%)	Ash (%)	Mass (%)	Ash (%)	Mass (%)	Ash (%)	Mass (%)	Ash (%)
Feed	100.0	54.7	100.0	57.7	100.0	55.3	100.0	57.6
Floats 1.80	37.2	29.2	33.2	30.3	34.8	28.0	31.5	27.6
Sinks 1.80	57.7	71.6	65.4	71.9	60.2	71.6	62.9	73.1
-0.5mm	5.4	46.3	1.4	43.0	5.0	47.8	5.6	53.2
Flotation Feed	42.6	31.4	34.6	30.8	39.8	30.5	37.1	31.5
Flotation Concentrate	31.3	23.1	18.5	20.0	22.4	19.8	25.9	21.3
Flotation Tailings	11.7	46.8	16.1	43.1	17.4	45.0	11.2	49.8

After flotation, a product with an ash content of between 20-23% was achieved, with yields of 18.5-23.1% being observed.

Graphical representations of the flotation rate tests are depicted in Figure 6.19 to Figure 6.22, which also include the effect of floating the -1.4mm fraction of the discard, and for June and August, the -212 μ m fraction. A line indicating the yields expected at an ash content of 14% was included on each graph, an summarized in

Table 6.15, which includes the yields from the flotation tests and in brackets, relative to the process feed.

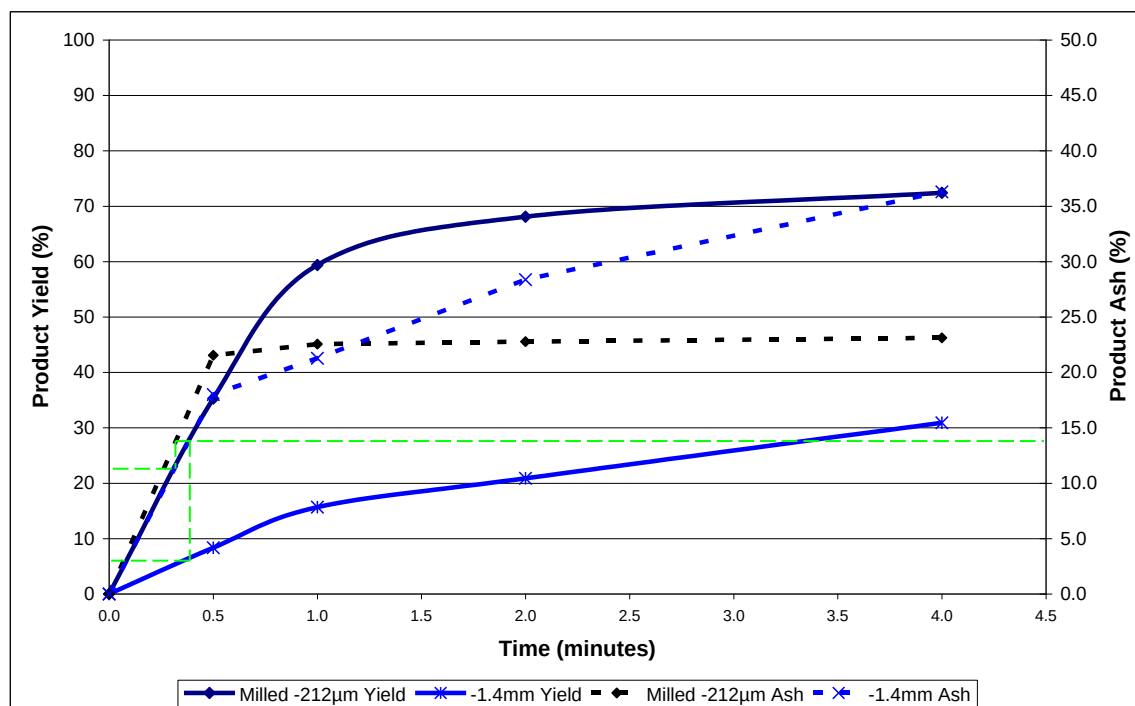


Figure 6.19: Flotation rate tests April 2008

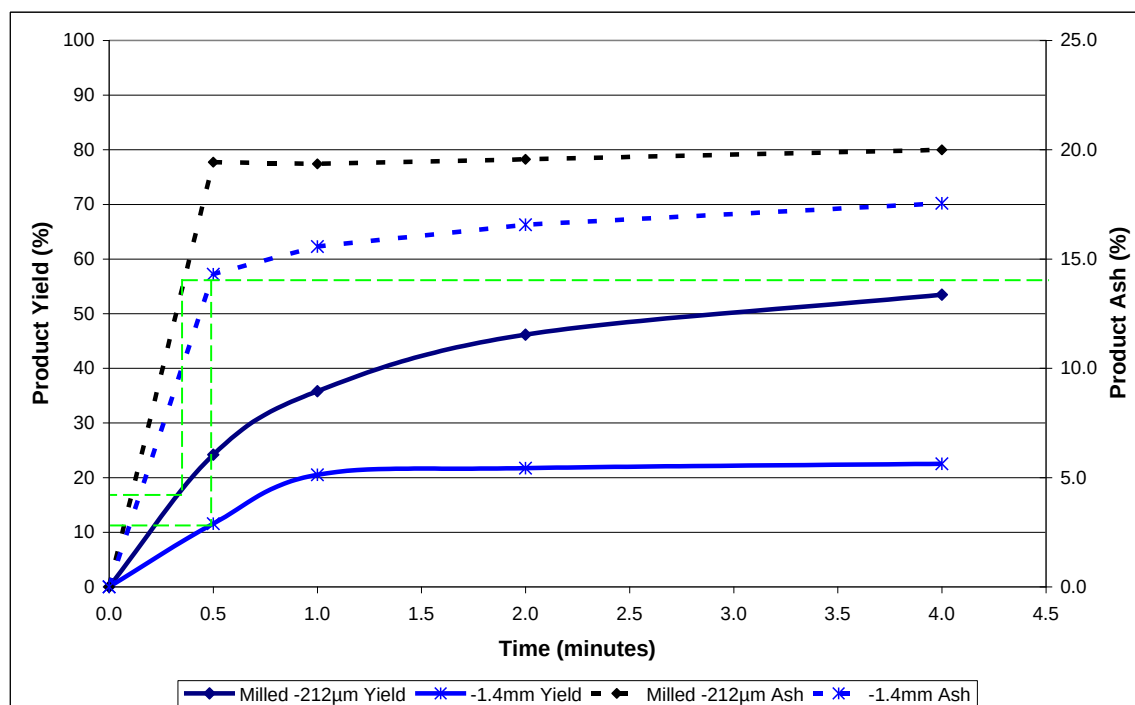


Figure 6.20: Flotation rate tests May 2008

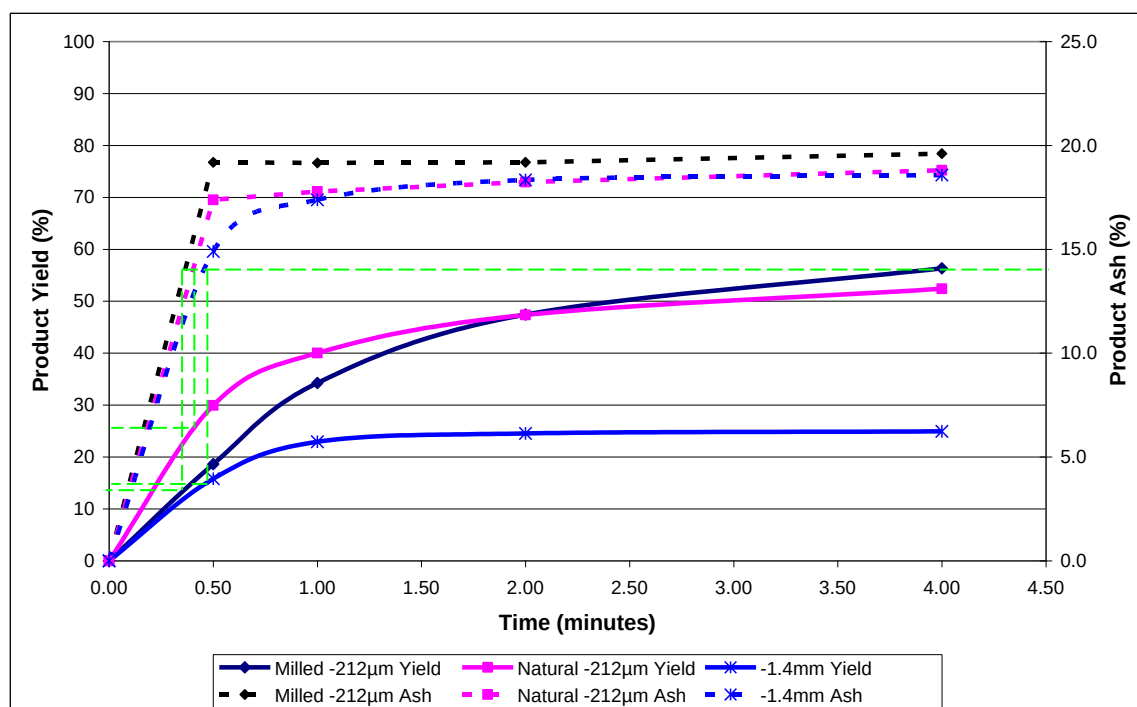


Figure 6.21: Flotation rate tests June 2008

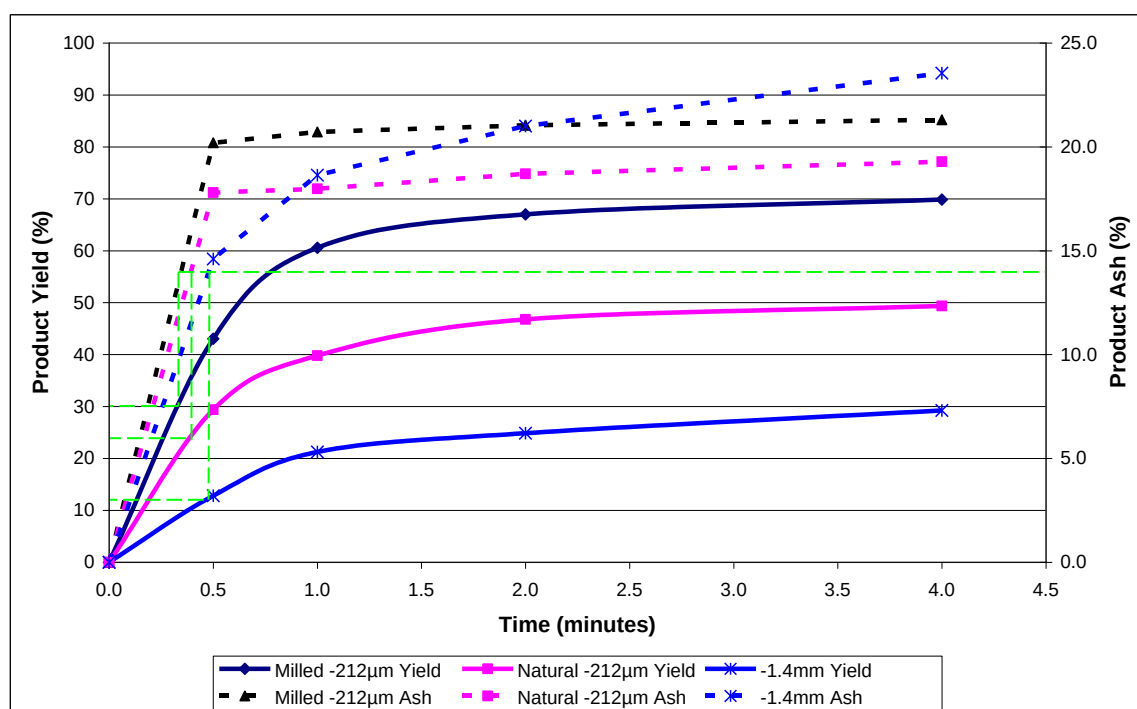


Figure 6.22: Flotation rate tests August 2008

Table 6.15: Summary of yields obtained from the flotation rate tests

Test	April 2008	May 2008	June 2008	August 2008
Milled to -212 μ m	22% (9.4%)	17% (5.9%)	14% (5.6%)	30% (11.1%)
Natural -1.4mm	6% (1%)	11% (1.9%)	15% (2.6%)	12% (2.1%)
Natural -212 μ m	-	-	26% (0.4%)	24% (0.6%)

After removal of the -0.5mm material, destoning the +0.5mm material and recombining the destoned product with the -0.5mm material and milling to -212 μ m and floating, it was observed from the information contained in graphs Figure 6.19 to Figure 6.22 that at 14% ash, a flotation product yield of between 14% to 30% could be expected, which equated to a yield of between 5.6% to 11.1% based on the overall discard, and just under half of this when calculated to a plant feed basis. (The actual plant yields for these four months were 53.1% for April 2008, 56.7% for May 2008, 56.1% for June 2008, and 56.4% for August 2008 respectively). At this point it could be concluded that, along with the results from the borehole testing, that the original concept of destoning-milling froth flotation type of process may not be an economically viable option for the retreatment of the Tshikondeni discard material, though no optimization of the flotation reagent suite had been considered to this point. Some optimization work was performed later in the study as detailed in sections 6.3.3.4 and 6.3.3.5 of this chapter.

The natural -1.4mm and -212 μ m comparative test indicated that milling improved the floatability of the plant discard. The relatively high yields obtained from the naturally generated -212 μ m fraction initially indicated significant losses in this stream, but once worked back to the overall discard feed, they were observed to be in the order of 0.5%. Working this back to the plant feed, a yield in the order of 0.2% would be achieved and is an indicator that the flotation section of the plant was working efficiently in terms of this size fraction.

The composition of the plant feed was examined in terms of the proportions delivered as run of mine from the three shafts and no correlation could be found regarding the effect of any shaft on the yields obtained.

6.3.3 Old Discard Dump – 20 Tonne Bulk Sample

6.3.3.1 Rougher Rate Tests

Having seen in the previous section (section 6.3.2) that the coarser particles of the arising discard contained little potential for successful upgrading, it was considered that removal of these particles ahead of destoning prior to milling may give a better potential to recover a coking coal product. The +8mm fractions were noted to have high raw ashes (>60%), whilst the -8+4mm and -4+2mm fractions were noted to be marginally better at 54-56% ash. Therefore 3mm was selected as a starting point. The destoning was also carried at a slightly lower separation relative density of 1.75 compared to 1.80 used on the borehole samples. Three rougher rate tests were performed using different reagent suites, and after removal of the analytical aliquots, the remaining concentrates were combined together prior to further milling and a cleaner flotation stage.

The three rougher rate test results are summarized in Table 6.16.

Table 6.16: Summary of rougher rate flotation tests

Test 1		320 g/t Paraffin and 50 g/t FTN3			
Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Conc 1	0.50	34.8	34.8	16.1	16.1
Conc 2	1.00	13.5	48.3	21.6	17.6
Conc 3	2.00	20.7	69.0	23.6	19.4
Tails		31.0	-	34.8	-
Feed		100.0	100.0	24.2	24.2
Test 2		160 g/t Paraffin and 25 g/t FTN3			
Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Conc 1	0.50	21.8	21.8	15.1	15.1
Conc 2	1.00	22.6	44.4	19.1	17.1
Conc 3	2.00	28.2	72.7	23.6	19.7
Tails		27.3	-	35.1	-
Feed		0.0	100.0	23.9	23.9
Test 3		100 g/t FTN3			
Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Conc 1	0.50	39.7	39.7	16.1	16.1
Conc 2	1.00	16.6	56.3	21.5	17.7
Conc 3	2.00	17.9	74.2	24.8	19.4
Tails		25.8	-	36.4	-
Feed		0.0	100.0	23.8	23.8

The three tests indicated high product yields varying between 69-74% and final concentrate ash contents in the order of 19%. The first concentrate, collected at 30 seconds also produced high ash contents in the order of 15-16%, the lowest being test 2 which used the equivalent of one quarter of the typical dosage used on fresh feed material entering the flotation circuit. A higher yield and associated mass would be expected from a laboratory test due to the confined volume and very high relative energy input.

6.3.3.2 Cleaner Rate Test

The cleaner rate test performed on the concentrates from all tests gave concentrate ashes of between 15.9-18.8% and significantly higher mass yields due to the effect of regrind milling of the rougher concentrate. The mass yields in brackets indicate the yield relative to the average rougher feed producing an average rougher concentrate yield of 72% calculated from Table 6.16 above.

Table 6.17: Cleaner rate test results

150g/t PAR and 300g/t Cytec S-8860 GL					
Stream	Time (min)	%Mass Yield	Cum. Yield	%Ash	Cum. % Ash
Conc 1	0.50	57.3 (41.3)	57.3 (41.3)	15.9	15.9
Conc 2	1.00	20.0 (14.4)	77.3 (55.7)	17.6	16.3
Conc 3	2.00	6.5 (4.7)	83.9 (60.4)	22.3	16.8
Tails		16.1	-	31.5	-
Feed		100.0	100.0	19.2	19.2

From the above data it was obvious that a lower mass yield would be required to meet the product specification of 14% ash and, based on the worked performed on the boreholes, it was suspected that the use of tribromoethane (TBE) and toluene used in the float and sinks work may have had an influence on the material surface properties, which is discussed further in the next section.

6.3.3.3 The Effect of Toluene and Tribromoethane on Flotation

The results of the two sets of tests as described in section 6.2.3.3 are summarized in Table 6.18 below. It is clearly seen that for the paired tests, that the material treated

with TBE and toluene was noted to produce a higher yield to flotation concentrate, which varied between 12-15%, and an associated increase in concentrate ash content of up to 2%.

It can therefore be deduced that chemicals such as TBE and toluene used to perform dense liquid separation ahead of froth flotation has a detrimental effect on the subsequent flotation test, and higher yields and ashes should be expected.

Table 6.18: Summary of the effect of TBE and toluene on flotation performance

Test	No TBE and Toluene		With TBE and Toluene	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)
Test 1 Concentrate	37.0	21.2	50.6	24.6
Test 1 Tailings	63.0	59.6	49.4	62.1
Feed	100.0	45.4	100.0	43.1
Test 2 Concentrate	45.0	24.3	50.7	24.8
Test 2 Tailings	55.0	62.9	49.3	58.2
Feed	100.0	45.5	100.0	41.3
Test 3 Concentrate	36.3	21.8	48.3	24.7
Test 3 Tailings	63.7	57.3	51.7	60.7
Feed	100.0	44.4	100.0	43.3

6.3.3.4 Flotation Tests after Destoning By Dense Medium Separation

The detailed results for this series of tests are included in Appendix 6 and the overall summarized results presented in Table 6.19. Tests were also performed on the -6mm fraction and the whole discard as a means of comparison.

Table 6.19: Summary of the flotation performance after destoning by dense medium separation

Sample	Test Conditions	Yield (%)	Ash (%)
-3mm Discard F1.75	150 g/t FTN4	43.4	16.7
	150 g/t Paraffin , 150 g/t FTN4	50.3	18.6
	150 g/t Paraffin , 150 g/t FTN4 , 300g/t Cytec S-8860 GL	14.1	15.4
-6mm Discard F1.68	100 g/t Paraffin , 50 g/t FTN4	41.4	23.9
	100 g/t Paraffin , 50 g/t FTN4 , 100g/t Cytec S-8860 GL	25.7	21.2
	150 g/t Paraffin , 50 g/t FTN4 , 100g/t Cytec S-8860 GL	30.3	22.1
Whole Discard F1.72	100 g/t Par , 100 g/t FTN4 (OLD CELL)	48.2	24.8
	200 g/t CRX 11	40.4	24.7
	400 g/t CRX 11 and 100g/t FTN4	49.1	25.3
	200 g/t CRX 11 and 200g/t FTN4	49.1	25.3
Whole Discard F1.72	200 g/t CRX 11+400g/t FTN4	38.8	19.5

From the results it can be seen that the lowest ash value achieved was 15.4% when using 150 g/t Paraffin , 150 g/t FTN4 and 300g/t of the ash depressant Cytec S-8860 GL, but the yield was extremely low with only 14.1 % of the material recovered. It was also found that the ash depressant helped in lowering the percentage ash of the flotation concentrate, with the ash content decreasing from 18.6% to 15.4% for similar conditions on the -3mm material. On the -6mm material, the ash content decreased from 23.9% to 21.2% when adding the ash depressant. Unfortunately, the yields also decreased substantially in each case, the effect being observed to be more pronounced for the -3mm material compared to the -6mm material

The third sample of the -3mm material, which used the Cytec S-8860 GL depressant was also submitted for basic coking coal testing which returned a free swelling index value of 5 and a Roga Index of 44. Though not on specification, it indicated that the flotation product contained some potential for use in a coking process, as either a blend feedstock along with “better” sources of coking coal or for production of market coke.

6.3.3.5 Flotation Test in the -0.5mm Size Fraction

The detailed results for this series of tests are included in Appendix 6 and summarized in Table 6.20

Table 6.20: Summary of the flotation performance of the -0.5mm material

Sample	Reagent Dosage (g/t)			Test Type	Yield (%)	Ash (%)
	PARAFFIN	CRX 11	FTN 4			
-0.5mm No Milling	200	200	400	Bulk Rougher Test	44.1	20.1
-0.5mm Milled to -212µm	200	400	400	Bulk Rougher Test	69.2	27.8
-0.5mm No Milling	0	100	100	Rougher Rate Test	33.2	15.3
-0.5mm No Milling	0	200	100	Rougher Rate Test	39.9	16.3
-0.5mm No Milling (Average of Results)	0	100	100	Bulk Rougher Tests x7	39.3	17.2
-0.5mm No Milling (Average of Results)	100	0	100	Bulk Rougher Tests x4	28.3	16.4
Rougher Concentrate Milled for 1 minute	100	0	100	Bulk Cleaner Test	54.1 (15.3)	13.2
Rougher Concentrate Milled for 2 minute	100	0	100	Bulk Cleaner Test	47.7 (13.5)	13.6
Rougher Concentrate Not Milled	50	0	50	Bulk Cleaner Test	80.5 (22.8)	13.9
Rougher Concentrate Not Milled	100	0	100	Bulk Cleaner Test	78.4 (22.2)	14.0

The first two tests, i.e. the first upon the -0.5mm material and the second on the -0.5mm material milled to 212µm showed potential with a yields of 44.1% and 69.2% with associated ash contents of 20.1% and 27.8% being achieved for the respective tests.

The next set of tests used less reagents to attempt to improve the product ash and generate material for possible cleaner tests. The rougher rate tests returned final concentrate yields and ashes of 33.2% and 15.3% for test 1, and 39.9% yield at 16.3% ash respectively, with the higher collector dosage returning the higher values. The set of seven repeat bulk rougher tests returned results similar to the 200g/t rougher rate test, whilst the four replicate tests using paraffin as the collector gave an average yield of 28.3% at an ash of 16.4%.

The paraffin concentrates were therefore used as the feedstock for the cleaner tests. Milling of the concentrate prior to cleaner flotation did not appear to improve the floatability of the material and the effect of reagents on the two non-milled samples did give almost identical yields (22.8% and 22.2% relative to rougher feed) and ashes at 13.9% and 14.0% respectively.

Though some reagent optimization was attempted in this and the previous section of the work, a more thorough investigation is required to maximize the yield achievable beyond the order of 22% at 14% ash. Whether a significant yield increase can be achieved is somewhat doubtful though, and the use of milling and froth flotation for such low yields would have a severe negative impact upon the process design from both capital and operational cost perspectives.

6.4 Chapter Summary

The initial phase of the test programme was undertaken on the auger borehole core samples after screening destoning and milling to $-212\mu\text{m}$. These samples were subjected to the standard Tshikondeni flotation rate test using the standard reagent suite comprising 85% collector (paraffin) and 15% frother (Betafroth BTN3), which was dosed at a relatively low dosage equating to approximately 0.7kg/t in the flotation circuit.

An average mass pull (yield) of 25.5% at an average ash content of 19.3% was observed for the old dump, whilst for the current dump the average mass pull and ash content were noted to be 18.8% and 20.3% respectively. The standard test

showed that little selectivity was experienced in the flotation test, and the first concentrate (0-30 seconds) contained most of the mass and ash contents above 15%. The high ash content could potentially be improved by reagent optimization and/or the addition of a cleaner stage. Several samples along with the plant feed and plant discard should have been subjected to a release analysis to gain a better understanding of the flotation process and eliminate the effects of entrainment.

The flotation rate information from application of the Klimpel and Kelsall flotation models indicated the dumps to be slow floating.

The samples from the Visgat Dam also produced high ash contents overall, though it was noticed that the lower ash material floated later in test, contrary to what would be normally expected. Again, tailoring of the process in terms of reagents and cleaner flotation may positively influence the behaviour of this material. The modelled flotation rate constants also indicated slow flotation characteristics

Tests were performed on the arising discard composites from four separate months and again the specification ash content was not reached and varied between 20-23% at yields of between 18.5-23.1%. Milling of the material appeared to marginally improve the overall yield obtained, but further work to optimize the process, particularly in terms of reagent selection would be required.

The 20 tonne bulk sample taken from the old discard dam was subjected to more extensive testing, and included the combined sizing followed by dense medium separation considered in previous chapters.

For the -3mm material, which was destoned at a slightly lower relative density of 1.75, yields of 69% to 74% were achieved in a rougher stage of flotation at ash contents of approximately 19%, using slightly different reagent suites. These rougher concentrates were then combined and subjected to a cleaner stage, which gave a yield of approximately 84% at an ash content of 16.8%.

It was noted that the use of organic liquids in the destoning process appeared to influence the flotation performance: i.e. lower product qualities and minimal response in terms of selectivity were observed. A short study comparing the use of organic

liquids to a suspension of magnetite and water was performed where it was found that the use of organic liquids positively affected the yield by between 12-15%, and increased the product ash by up to 2%.

Further work was therefore undertaken on the discard at various sizes and destoning relative densities and alternative reagent suites, where it was noted that the removal of the finer sizes (-6mm or -3mm) would give improved concentrate ashes, though lower yields. Better results could possibly be achieved by further testwork to optimise the destoning and flotation test programmes.

On the -0.5mm fraction only, it was observed that concentrate ashes of close to the 14% specification could be achieved from cleaning the rougher concentrate, but at fairly low overall yields.

6.5 Chapter References

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Chapter 7: Liquid-Solid Separation and Briquetting

This short chapter examines liquid-solid separation and briquetting as may be applied to the production of a froth flotation concentrate generated from the froth flotation product made after destoning and milling the discards.

A short literature survey is presented covering both topics both generally, and as applicable to Tshikondeni. Thereafter the very limited work performed on the froth flotation concentrate produced in the previous chapter is presented. Competent briquettes were able to be produced, but not tested due to limitations in terms of mass available for bulk briquette production.

7.1 Literature Survey

Having produced a product from the flotation stage, the next step is to reduce the moisture content to an acceptable level. However, given that the flotation product will be very fine, it is also critical that it can be handled effectively which would possibly entail the use of a briquetting process.

7.1.1 Liquid Solid Separation

Liquid solid separation in relation to coal preparation was well documented in a paper written by Du Preez in the early 1980s (Du Preez, s.a.). In this work he indicated that the removal of water is a direct function of the particle size, and that the finer the particle size, the greater the energy required in removing the moisture, the more costly the equipment required and the higher the operating costs. He summarized the moisture content of the various size products, after natural drainage for varying lengths of time as a graph presented in Figure 7.1.

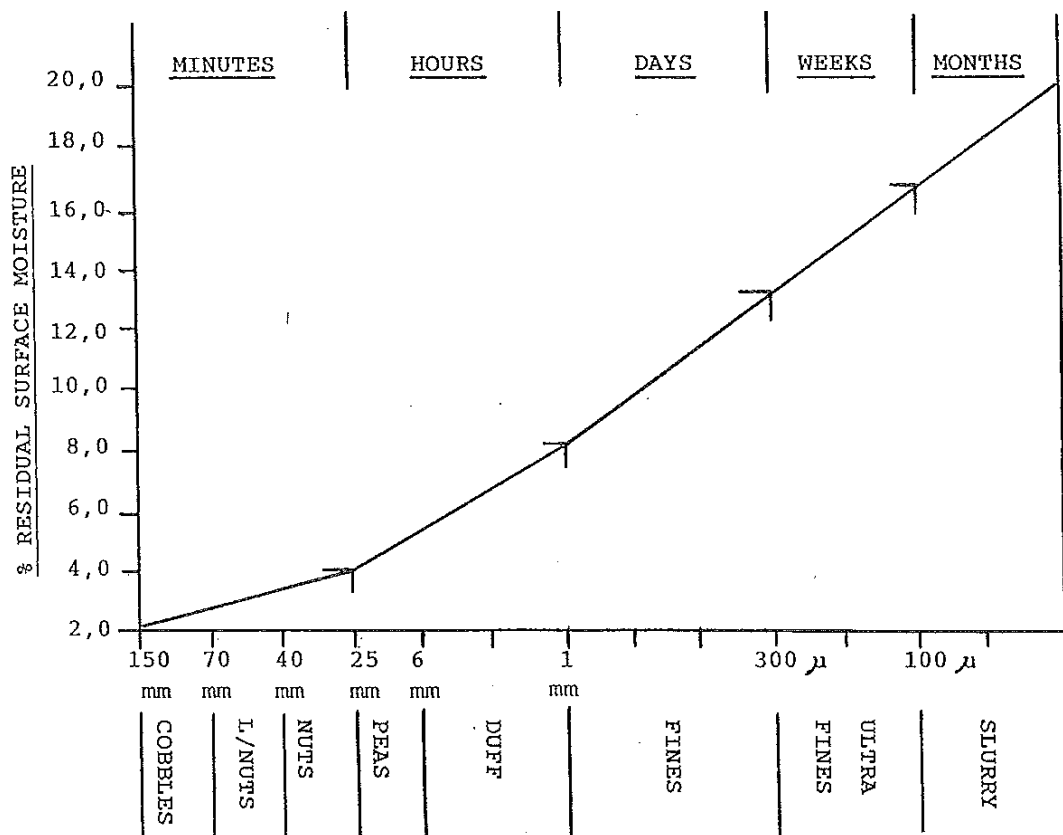


Figure 7.1: Residual surface moisture vs. particle size (Du Preez, s.a.)

Du Preez then identifies the various types of equipment available for dewatering the different size products as illustrated in Figure 7.2. Certain types of equipment function better than others in any given size range, and that some overlap between the various pieces of equipment is observed.

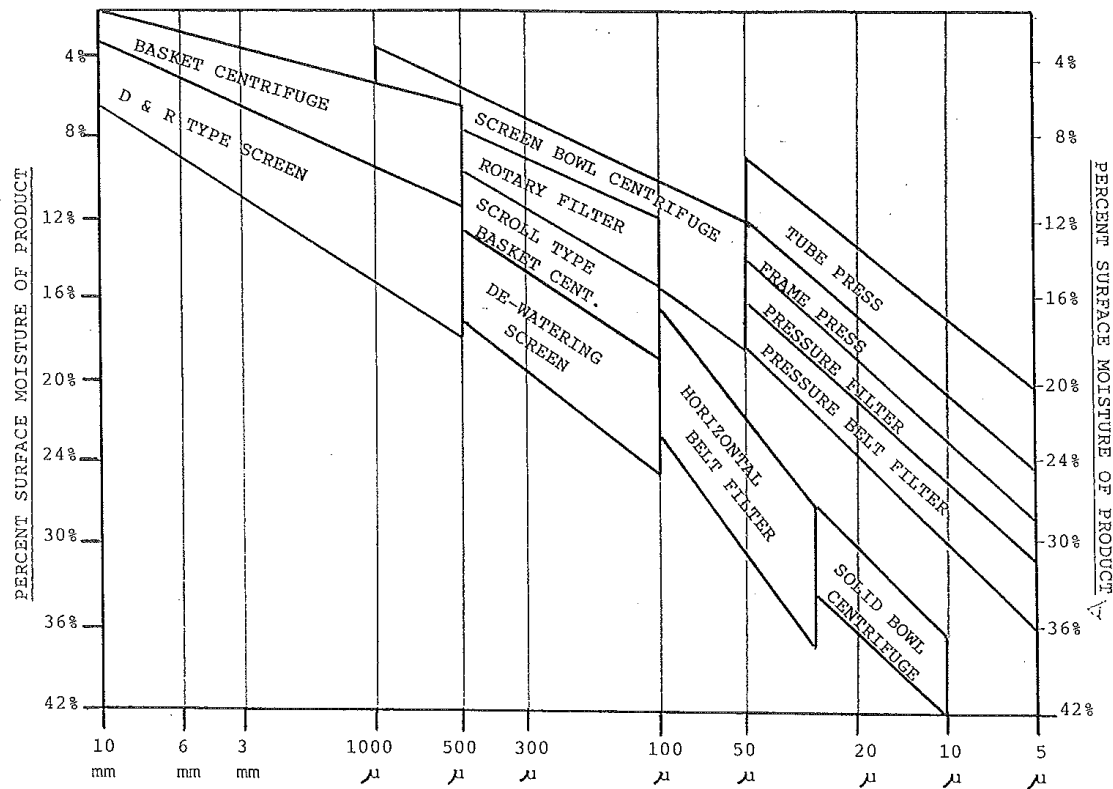


Figure 7.2: Product surface moisture vs. particle size range (Du Preez, s.a.)

The existing plant at Tshikondeni uses solid bowl centrifuges to dewater the flotation plant product and have sufficient capacity to handle a significant increase in feed produced by the proposed flotation section of the discard retreatment plant. The solid bowl is particularly suited to this duty since it is able to handle flotation product in the range -1.4+0mm. Other methods of dewatering are therefore not discussed further in this section. A typical solid bowl centrifuge is illustrated in Figure 7.3 (taken from the Flottweg Catalogue, courtesy Kottman 2007)

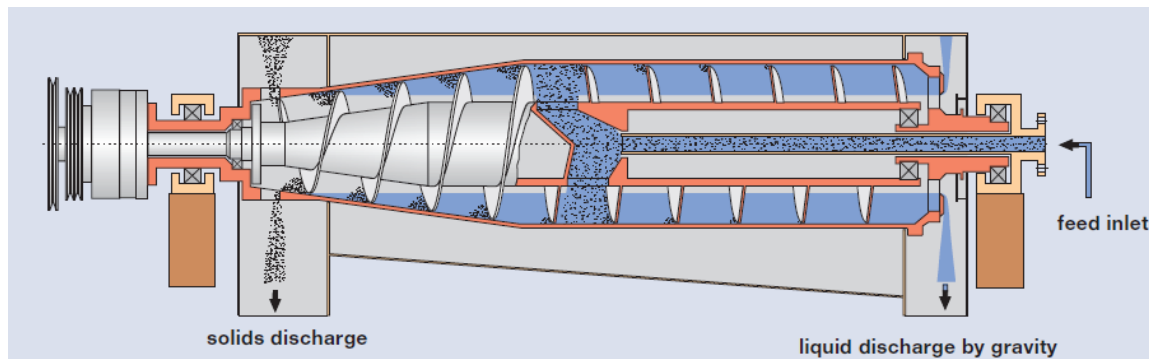


Figure 7.3: Solid bowl centrifuge (Kottman, 2007)

The unit consists of a horizontally revolving cylindroconical bowl, inside which a screw conveyor of similar section rotates in the same direction at a slightly higher or lower speed. The feed pulp is introduced through a centre pipe, which on leaving the pipe is subjected to high centrifugal force causing the solids to settle on the inner surface of the bowl at a rate dependent upon the rotational speed employed (typically 1600-8500 rpm). The separated solids are conveyed out of the liquid by the scroll and discharged through the outlets at the smaller end of the bowl. The solids are continually dewatered by the centrifugal force as they move from the feed end to the discharge end. Excess entrained liquids drain away to a pond circumferentially through the particle bed and drains out of the wider end of the bowl.

Pretorius (2006) undertook an internal Tshikondeni investigation into the conversion of the screen bowl centrifuges to solid bowl centrifuges at the plant to prevent losses from the $-45\mu\text{m}$ fraction of the flotation product. This was successfully performed and the plant saw an increase in yield of approximately 3%, though the moisture of the product was slightly elevated at between 22-24%. This was found to be still acceptable since the final product lies on the product stockpile for 2 to 3 days prior to loading and meets the required product specification.

Pretorius (2006) also noted that the design capacity of each of the two centrifuges was 32t/h compared to the 15t/h that they typically receive from the flotation section of the plant. Therefore, these units would most likely have sufficient capacity to treat an additional flotation product produced if the plant discards are treated by the proposed destoning-milling-froth flotation process described in the preceding chapters.

7.1.2 Briquetting

Once the ultra-fine coal product from the froth flotation section has been dewatered, it still has a relatively high moisture content (>20%), making it sticky and creating handling problems such as blockages in chutes etc. Moreover, if too dry it would be easily blown away by a strong wind. One way to overcome this is to process the dewatered material further using a briquetting or pelletizing technique. Mangena (2001) did an extensive study on the use of binderless briquetting on several South African coals summarized in Table 7.1, with the requirement that a minimum dry compressive strength of 350kPa was reached, all samples attained this criteria, and the samples were tested on both laboratory and pilot plant scale thereby proving the economic viability of the concept.

Table 7.1: Summary of binderless briquetting results (Mangena, 2001)

Coalfield	Colliery	Product	*Dry Compressive strength (kPa) 90 % CL	Wet Compressive strength/Water resistance (kPa) 90 % CL
Witbank	Optimum	Slurry	515 ± 29	0
Witbank	Koornfontein	Slurry	510 ± 49	98 ± 8
Witbank	Bank No.2 Seam	Thickener underflow	581 ± 40	145 ± 11
Witbank	Optimum PSS	Thickener underflow	678 ± 46	0
Witbank	Kleinkopje	Flotation	661 ± 40	140 ± 13
Witbank	Bank No.5 Seam	Spiral	519 ± 38	119 ± 10
Witbank	Leeuwpans	Slurry	681 ± 37	0
Waterberg	Grooteegeluk	Spiral (coking)	779 ± 39	432 ± 25
Soutpansberg	Tshikondeni	Flotation (coking)	1112 ± 90	694 ± 31

*Minimum required is 350 kPa

It was noted that the briquettes produced from the flotation product of the Tshikondeni plant gave the best results in terms of both the dry compressive strength and wet compressive strength (Water resistance) at 1112±90kPa and 694±30kPa respectively, and this was attributable to the high vitrinite content. Water resistance was also attributed to the high vitrinite content as well as the presence of the flotation collector and lower ash content relative to the other samples.

Mangena (2007) reported that there are several ways of producing briquettes or pellets as illustrated by means of a roller press or a pellet mill depicted in Figure 7.4 and Figure 7.5 respectively. The roller press allows the fine material to be fed either

gravitationally or by a screw feeder (1), between two oppositely turning rollers (c) under a suitable applied pressure (b).

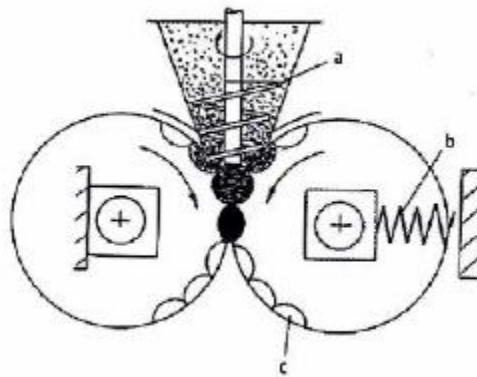


Figure 7.4: Operation of a roller press with a screw feeder (Mangena, 2007)

The pellet mill extrudes a pasty feed material through a die plate and cut with a blade to the desired length. As can be seen in Figure 7.5, the feed material is introduced into the space between a rotating perforated ring and solid rollers (A). It is then compressed and forced through cylindrical openings to form the extrudates/pellets (B) which are then cut to length by the adjustable blades (C) on the outer side of the ring.

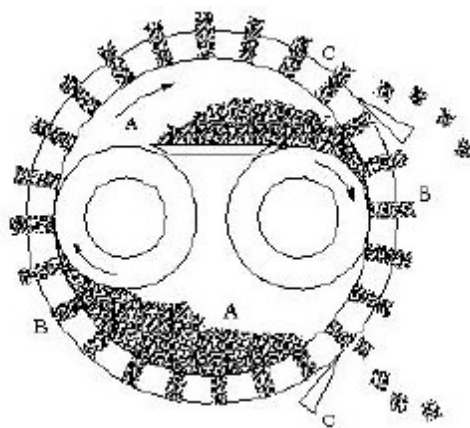


Figure 7.5: Operating principle of a pellet mill (Mangena, 2007)

In terms of application of this type of technology in South Africa, McMillan (2002) indicated that the Carbolite[®] process, which used starch as a binder, and renamed the Kkarbolite[®] process (due to being introduced at Klienkopje Colliery, commonly referred to as “KK”), had been modified to include only six of the original nine steps

in the process, i.e. static de-lumping, binder dosing, mixing, de-lumping, pelletizing and finally rotary screening to remove <2mm particles and drying at a temperature of approximately 150°C to give a total moisture of around 3%.

The cost of the binder was found to account for approximately 79% of the operating costs, and would have a negative impact on the project economics given the relatively low value of steam coal. The process is illustrated in Figure 7.6.

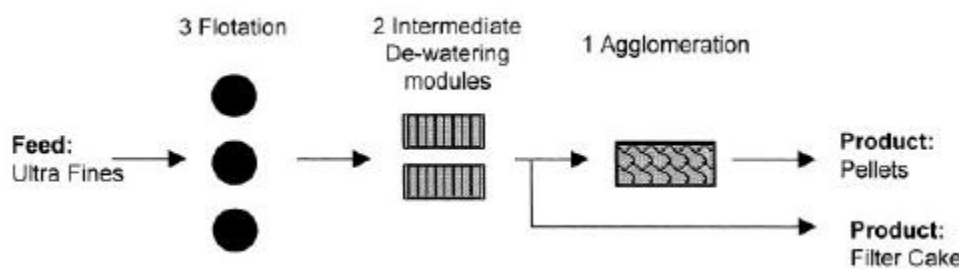


Figure 7.6: the Kkarbolite process (McMillan, 2002)

This type of technology may have been applicable at Tshikondeni given the good response of the coal to a binderless briquetting process and the higher prices attainable for a hard coking coal compared to steam coal.

7.2 Briquetting Experimental Test Work and Results

Having produced a flotation concentrate from a bulk flotation test as described in Section 6.2.2 above, a laboratory scale briquetting test was performed on the flotation concentrate at 5% moisture. No binder was used, since Mangena (2001) had established that the Tshikondeni fines were very amenable to binderless briquetting. As can be seen from Figure 7.7, the dried sample was not easy to re-wet as expected from Mangena (2001) due to the presence of flotation reagents on the surface and potentially a high vitrinite content.



Figure 7.7: Flotation product with water addition prior to mixing and briquetting

The apparatus used to form the briquettes is shown in Figure 7.8, whilst Figure 7.9 shows a typical briquette as formed by the apparatus.



Figure 7.8: Bench scale briquetting apparatus

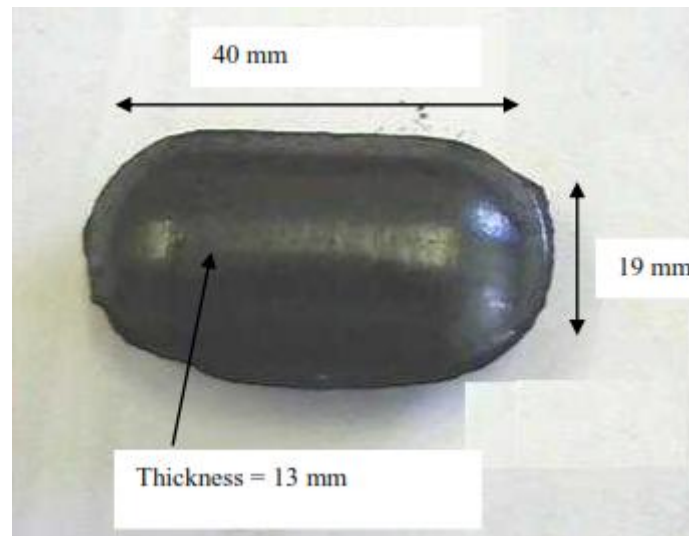


Figure 7.9: Visual illustration of the briquette formed by the laboratory apparatus

With sufficient material only being available to produce a few briquettes, these were not tested for compressive strength since they would not be representative of the expected full scale plant. The few briquettes produced though were, however, noted to be competent.

7.3 Chapter Summary

After the destoning, milling and flotation of the discard, the flotation product would need to be dewatered, and due to the fine size, most likely require briquetting to enable ease of handling.

The existing solid bowl centrifuges are capable of dewatering the froth flotation product effectively, and the existing plant potentially has surplus capacity which could potentially be used to dewater the froth flotation product from a discard retreatment plant.

Tshikondeni coal is very amenable to binderless briquetting, and the application of this technique may be required if a discard retreatment plant is installed to enable the additional froth flotation product to be easily handled.

7.4 Chapter References

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Chapter 8: Proposed Plant Design and Economic Evaluation

Based on the work conducted in the previous section it is possible that a saleable product could be made from the discard streams, though a more extensive test campaign would be required to optimise the process with specific reference to the flotation circuit.

A plant design was conceptualized applying all aspects previously discussed in the thesis, and an economic evaluation of the project undertaken. Some parts of the financial evaluation could not be presented since it would illustrate in detail the internal Exxaro macroeconomic forecasts and project evaluation processes which are confidential.

8.1 Introduction

8.1.1 Introduction

Based on the work in the preceding chapters it was found that:

1. Removal of high ash material from the discards could be achieved by screening at 8mm.
2. The -8+0.5mm size fraction could be further destoned successfully by the application of dense medium separation at a relative density of 1.80.
3. The destoned -8+0.5mm product combined with the -0.5mm would be combined to form feed for the milling process to produce a flotation feed of -0.212mm.
4. Froth flotation could potentially produce an on specification product of 14% ash, though more optimization of the process would be required.
5. The flotation production would need to be dewatered, and the dewatered product formed into briquettes to allow for ease of handling from the product stockpile to the end use in the coke ovens at ArcelorMittal Van Der Bijl Park.

Therefore a high level design for the discards retreatment plant was conceptualized by Powell and Botha (2009) and is discussed further in the following section.

8.2 Proposed Plant Design

A high level plant design was developed and proposed by Powell and Botha (2009) to process approximately 5.6 million tonnes by taking into account the tonnages that it would be expected to process:

- Old discard dump approximately 2.5 million tonnes
- New/current dump approximately 1.8 million tonnes
- Visgat dam of approximately 27.5 thousand tonnes, but this could potentially be better used for supply to the two local brickmaking facilities

- Estimated discard to the end of life of mine approximately 1.3 million tonnes as per mine planning figure

To reduce capital expenditure, it was considered that the discards retreatment process should begin once all the operational shafts had been mined out and modifications to the current circuit made as required. This would also extend the life of the plant and allow for the retention of fifteen plant personnel since they are familiar with the plant operation. Furthermore, they may not have been able to find new positions within the Exxaro group or nearby communities.

The proposed plant design is presented in Figure 8.1, which as described earlier shows an initial screening stage to remove the high ash +8mm material, the -8mm material being fed onto a desliming screen to remove the -0.5mm material. The -8+0.5mm material is then destoned in a dense medium cyclone and the product fed into a ball mill to be milled to -0.212mm prior to being combined with the -0.5mm material from the desliming screen underflow and subjected to froth flotation. The flotation product is then dewatered using a filter press ahead of being briquetted prior to placing on the product stockpile.

The +8mm material and the -8+0.5mm dense medium cyclone stream will be deposited on a new discard dump, whilst the flotation tailings go through the thickener and into the tailing disposal dam.

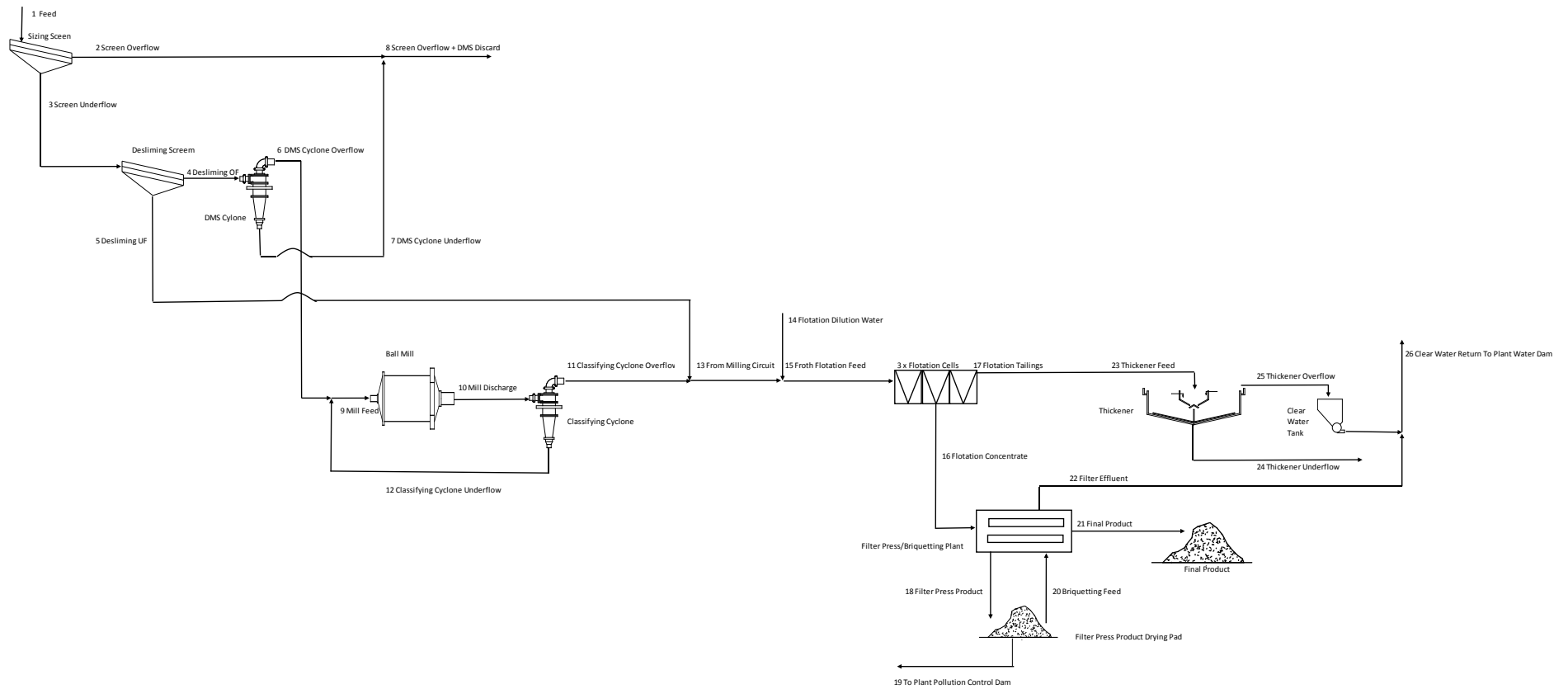


Figure 8.1: Proposed flowsheet for the Tshikondeni discards retreatment plant

The existing plant at Tshikondeni could be converted to process the discards with some modifications:

1. The ROM section which would require the removal of the scalping screen and installation of the screen to remove the +8mm material.
2. Decommissioning of the two dense medium cyclone section, which contain 710mm cyclones. As was demonstrated in section 2.2.2 these break away at around 2.8mm and therefore smaller units would be required. Consideration could be given to the use of a single 610mm unit operated at a higher than usual pressure of 11D (to give a better separation) or a single 660mm unit operated at the standard 9D pressure (Bekker 2009). It was also assumed that a yield from the dense medium stage of 16% could be realised, which is slightly higher than that presented in Chapter 4.
3. The installation of a ball mill to generate the -0.212mm flotation feed after destoning the -8+0.5mm dense medium cyclone product. The mill was sized at 5m long with a 3m diameter (2.79m diameter not available as per calculation) using the equation in Kelly and Spottiswood (1982) shown as equation 8.1 with the associated calculation presented in Table 8.1. The Mutale Plant Waste required mill power (6.99kWh/t) was used since this was the highest specific energy input reported in Chapter 5 (refer to Table 5.5). The mill was assumed to be a diaphragm discharge, hence the 1.13 value used for the mill type factor, the mill loading would be 35% and a critical speed factor of 0.20 was also used.

$$P = 8.44D_m^{2.5}LK_{mt}K_LK_{sp} \quad \text{Equation 8.1}$$

Table 8.1: Mill sizing calculation based on equation 8.1 (Kelly and Spottiswood, 1982)

Mill Sizing Calculation		
K_{mt}	mill type factor	1.13
Loading	(%)	35.00
K_L	loading factor	4.70
Critical Speed	(%)	80.00
K_{sp}	critical speed factor	0.20
P_s	kWh/t	6.99
T	Tonnage/hr	80.00
P (work)	kWh	559.20
$D_m^{2.5}L$	$m^{3.5}$	62.38
Length	m	5.00
Diameter	$m^{2.5}$	12.48
Diameter	m	2.79

4. The flotation, filter press, briquetting, thickening section was reviewed and found acceptable by Kruger (2009), whereby it was considered that 3 flotation cells of the current plant could be used, though it was suggested that the filter press and briquetting equipment be supplied by Kruger (2009). The existing thickener would be usable.

Kruger (2009) also remarked that more flotation testwork would be required to optimize the process and that milling of the -0.5mm material may also need to be considered if the project was to go into the detailed design phase. Furthermore, consideration could also be given to the installation of the Enprotec Dual flotation, which is illustrated in Figure 8.2.

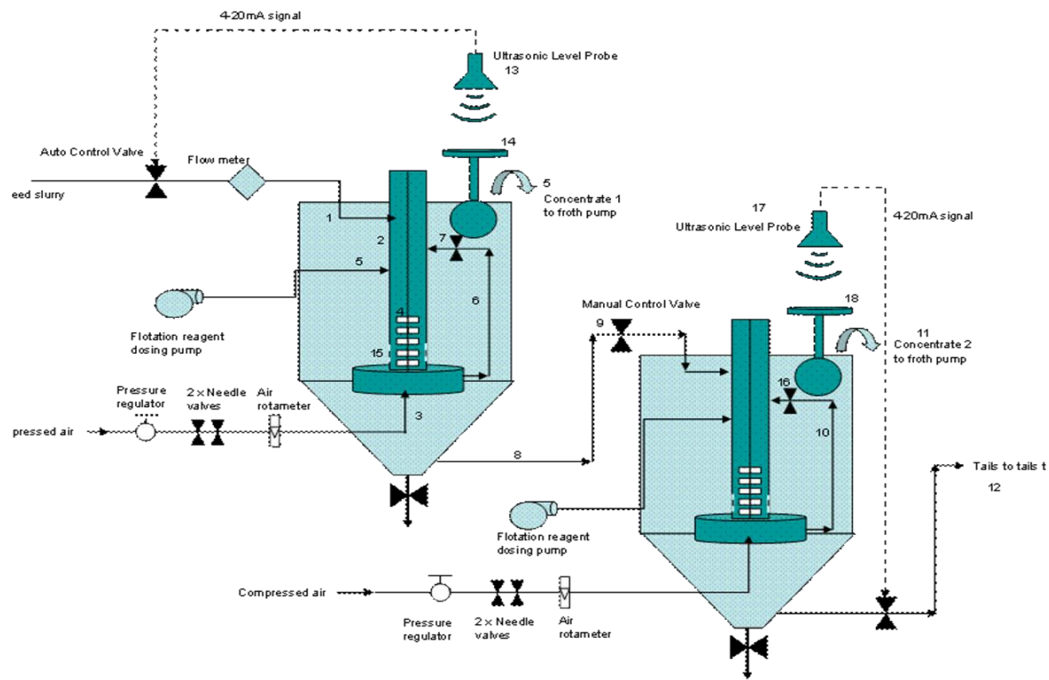
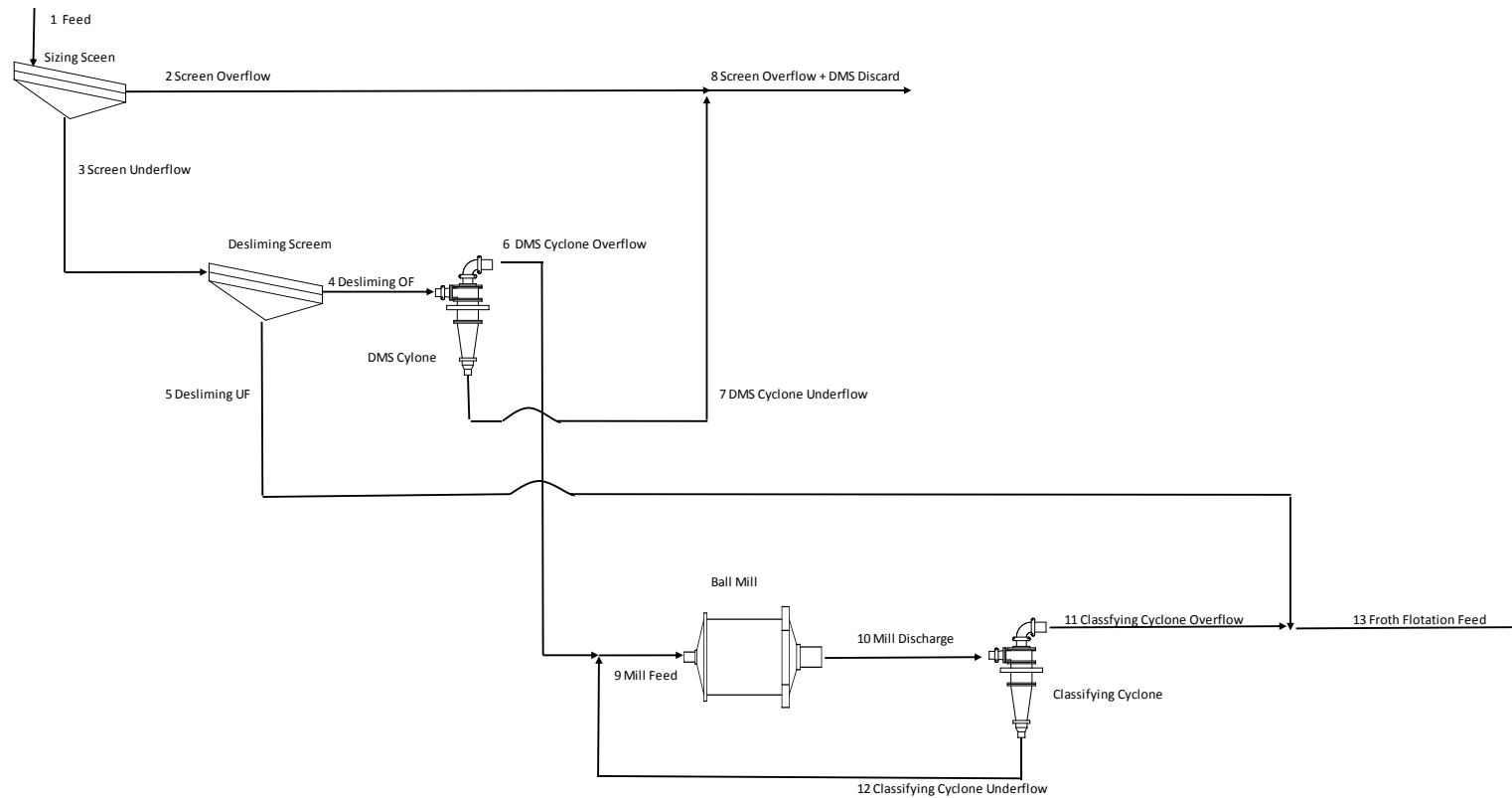


Figure 8.2: Enprotec Dual Flotation Cell (Kruger, 2009)

The benefits of the Dual Cell over other cells include vertical spindle pump with a tapered shaft for better air dispersion, diamond shaped holes in the shearing plates for better dispersion, micro bubble creation and high carrying capacities. The units installed to date have been found to give lower reagent and energy consumptions (Jacobs 2014).

The mass and water balances for the plant are illustrated in Figure 8.3, which considered the destoning and milling processes, and Figure 8.4 which considered the flotation, briquetting, thickener and final product section of the plant. The minor differences in terms of the tonnages and flow rates in Figure 8.4 are acknowledged and would be corrected during the detailed plant design: it should be borne in mind that this section of the plant design is somewhat optimistic in that it was designed with a 60% mass yield to concentrate at a 14% ash, which could not be achieved during the flotation test phase of the project. However, with optimization of the flotation process combined with the high vitrinite content of the flotation feed, it would be expected that 60% mass yield at 14% ash should be achievable.

Chapter 8: Proposed Plant Design and Economic Evaluation



	1	2	3	4	5	6	7	8	9	10	11	12	13
	Feed	Sizing Screen Overflow	Sizing Screen Underflow	Desliming Screen Overflow	Desliming Screen Underflow	DMS Cyclone Overflow	DMS Cyclone Underflow	Screen + DMS Cyclone Discard	Ball Mill Feed	Ball Mill Discharge	Classifying Cyclone Overflow	Classifying Cyclone Underflow	Froth Flotation Feed
Solids tph	80	56	24	23	1	13	9	65	46	46	13	33	14
Solids %	77	96	52	87	7	86	92	96	95	95	86	99	41
Water tph	24	2	22	3	18	2	1	3	2	2	2	6	20
Ash %	59	67	41	42	18	24	69	68	37	37	25	41	24
Size mm	-13+0	-13+8	-8+0	-8+0.5	-0.5+0	-8+0.5	-8+0.5	-13+0.5	-8+0.5	~-0.212	-0.212	+0.212	-0.5

Figure 8.3: Proposed flowsheet for the Tshikondeni discards retreatment plant: destoning and milling section

Chapter 8: Proposed Plant Design and Economic Evaluation

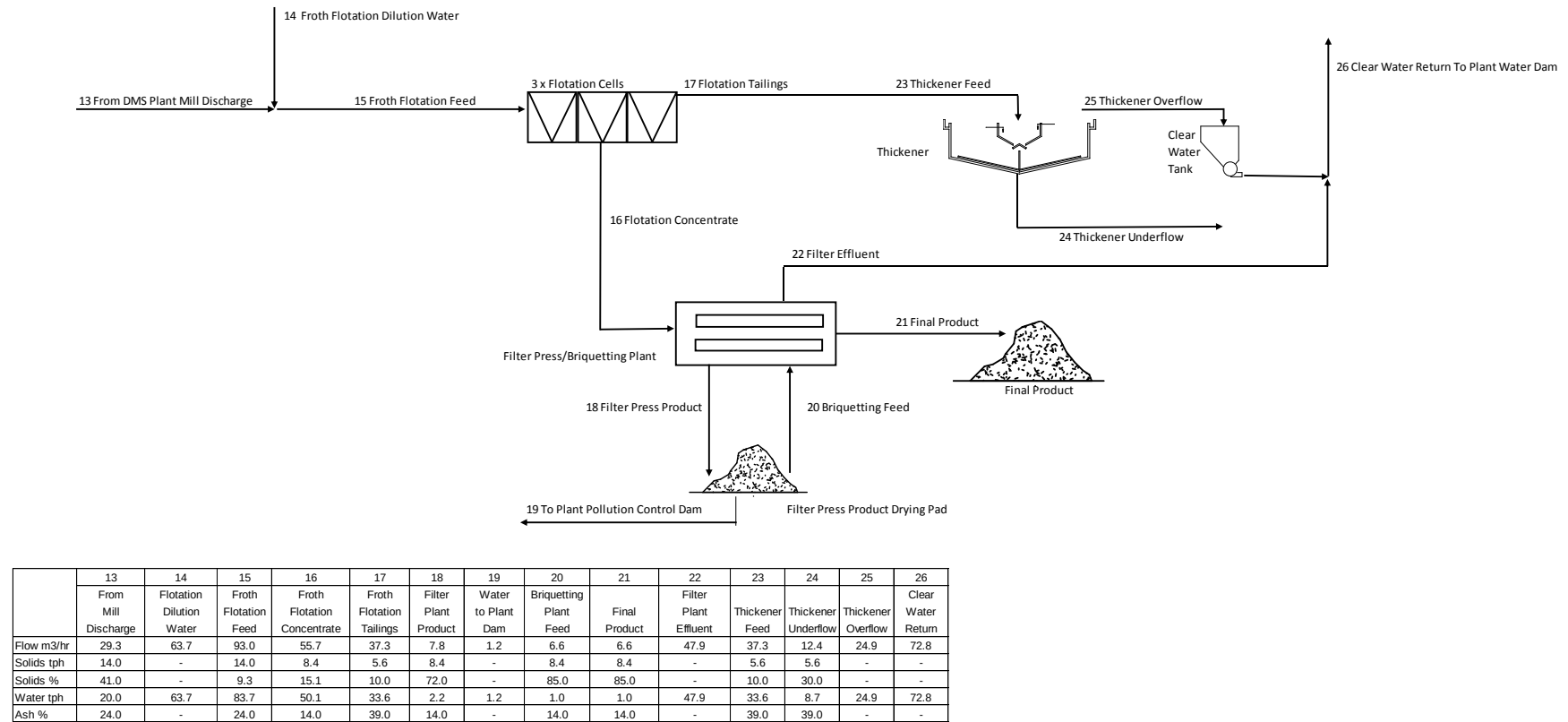


Figure 8.4: Proposed flowsheet for the Tshikondeni discards retreatment plant: flotation to final product section

8.3 Economical Evaluation

8.3.1 Plant Assumptions Overview

8.3.1.1 Plant Sizing

Based on the estimated 5.6 million tonnes of discards available, a plant operating at 80 tonnes per hour should be able to process the dump over a 10 year period based on the following parameters (Powell and Botha 2009), once the ROM sources has been exhausted:

- Plant operates 24 hours per day and 365 days per year, giving a potential to operate 8760 hours per year
- Availability and utilization was estimated to be 80%
- Therefore, the actual operating hours would be 7008 (80% of 8760 hours possible)
- At a plant feed rate of 80 tonnes per hour, the plant throughput would be 560 640 tonnes per annum (80tph multiplied by 7008 hours)

8.3.1.2 Capital Expenditure Estimates

The existing plant can largely be used to retreat the discards, but some modification of the circuit would be required as discussed in Section 8.2 of this chapter. These have been taken from Powell and Botha (2009) and summarized in Table 8.2.

Table 8.2: Estimated capital expenditure to modify the Tshikondeni plant for the retreatment of discards (Powell and Botha, 2009)

Capital Item	Cost (R)
Ball mill	1 157 625
Dewatering cyclone	80 000
Filer Press and Briquetting Plant	13 736 437
Screen	231 525
Sub Total 1 (Equipment)	15 205 587
Civil and Construction	3 000 000
Sub Total 2 (Equipment, Civils)	18 205 587
Contingency (20% of Sub Total 2)	3 641 117
Total	21 846 704

8.3.1.3 Working Costs Estimates

The estimated working costs for the retreatment plant were determined by Powell and Botha (2009) based on the prevailing Exxaro macro-economic values at the time and are summarised below:

- A total complement of 15 people, with 5 people per shift, would be required to operate the plant. At a typical cost to company of R12 000 per person per month, the labour costs would be R2 160 000 per annum.
- Electricity consumption was estimated at 150kWh, equating to 1.05MW/year, which at R0.55 per kW would cost R578 160 per annum.
- Water requirements were estimated at 80m³/hr, equating to 560 640m³/year, which at R7.60 per m³ would cost R4 260 864 per annum.
- Maintenance of the plant was estimated at R546 168 per annum and was based on 3% of the capex i.e. R18 205 587 x 3%.
- Transport costs to the Musina siding were estimated at R90/tonne, and at a 10% yield from the discard would require the movement of 56 064 tonnes (560 640 tonnes x 10%) at a cost R5 045 760 per annum (R90 x 56 064).

8.3.2 Macro-Economic Assumptions

The official company corporate macroeconomic forecasts and guidelines used at the time of the original study in Powell and Botha (2009) are presented in Table 8.3, which was extracted from Exxaro's Corporate Finance Department's Project Evaluation Tool.

Table 8.3: Macroeconomic assumptions used based upon the company corporate finance forecasts and guidelines in 2009(Powell and Botha, 2009)

Official Macro Assumptions Used	Source	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Escalation Factors/Rate																		
Annual																		
RSA CPI	Corporate Finance Forecast	7.0%	8.0%	6.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%
RSA PPI	Corporate Finance Forecast	7.5%	9.0%	7.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%
US Inflation	Corporate Finance Forecast	-2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%	2.0%
Manpower Inflation Forecast	Corporate Finance Forecast	10.0%	8.5%	6.0%	6.0%	6.0%	6.0%	6.0%	6.0%	6.0%	6.0%	6.0%	6.0%	6.0%	6.0%	6.0%	6.0%	6.0%
Electricity Increases	Corporate Finance Forecast	29.4%	26.3%	25.0%	13.8%	6.3%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%
Capex Escalation	Corporate Finance Forecast	12.0%	9.0%	7.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%
Other Operating Costs	Corporate Finance Forecast	9.5%	8.5%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%
Used in Model																		
RSA CPI	Corporate Finance Forecast	7.00%	8.00%	6.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
RSA CPI Factor	Corporate Finance Forecast	1.00	1.08	1.14	1.20	1.26	1.33	1.39	1.46	1.53	1.61	1.69	1.78	1.86	1.96	2.06	2.16	2.27
RSA PPI	Corporate Finance Forecast	7.50%	9.00%	7.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
RSA PPI Factor	Corporate Finance Forecast	1.00	1.09	1.17	1.22	1.29	1.35	1.42	1.49	1.56	1.64	1.72	1.81	1.90	1.99	2.09	2.20	2.31
SA inflation factor	Corporate Finance Forecast	1.00	1.08	1.14	1.20	1.26	1.33	1.39	1.46	1.53	1.61	1.69	1.78	1.86	1.96	2.06	2.16	2.27
Manpower Inflation Forecast	Corporate Finance Forecast	10.00%	8.50%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%	6.00%
Manpower Inflation Factor	Corporate Finance Forecast	1.00	1.09	1.15	1.22	1.29	1.37	1.45	1.54	1.63	1.73	1.83	1.94	2.06	2.18	2.31	2.45	2.60
Electricity Increases	Corporate Finance Forecast	29.38%	26.25%	25.00%	13.75%	6.25%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Electricity Escalation Factor	Corporate Finance Forecast	1.00	1.26	1.58	1.80	1.91	2.00	2.10	2.21	2.32	2.43	2.56	2.68	2.82	2.96	3.11	3.26	3.43
Capex Escalation	Corporate Finance Forecast	12.00%	9.00%	7.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Capex Escalation Factor	Corporate Finance Forecast	1.00	1.09	1.17	1.22	1.29	1.35	1.42	1.49	1.56	1.64	1.72	1.81	1.90	1.99	2.09	2.20	2.31
Other Operating Costs	Corporate Finance Forecast	9.50%	8.50%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%
Other Operating Costs Escalation factor	Corporate Finance Forecast	1.00	1.09	1.14	1.20	1.26	1.32	1.38	1.45	1.53	1.60	1.68	1.77	1.86	1.95	2.05	2.15	2.26
Exchange Rates																		
Annual																		
R/\$, nominal	Corporate Finance Forecast	9.00	9.10	9.19	9.29	9.56	9.84	10.13	10.43	10.74	11.05	11.38	11.71	12.06	12.41	12.75	13.15	13.54
Used in Model																		
R/\$	Corporate Finance Forecast	9.00	9.10	9.19	9.29	9.56	9.84	10.13	10.43	10.74	11.05	11.38	11.71	12.06	12.41	12.75	13.15	13.54
Tax Rate																		
RSA Company Tax Rate	South African Revenue Services	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%	28%
Commodity Price																		
Hard Coking Coal	Corporate Finance Forecast	120	137	154	180	149	137	129	128	128	128	128	128	128	128	128	128	128
Discount Rate																		
Discount rate	Corporate Finance Guideline	23.0%																
WACC, used in model	Corporate Finance Guideline	18.0%																
WACC, nominal	Corporate Finance Guideline	18.0%																
Technical Risk Premium	Corporate Finance Guideline	1.0%																
Project Phase Premium (Pre-Feasibility)	Corporate Finance Guideline	4.0%																

8.3.3 Detailed Economic Analysis

8.3.3.1 Original Study Analysis

Using the information presented in Section 8.3.1 and 8.3.2, a detailed economic analysis was undertaken and is presented in Table 8.4, Table 8.5 and Table 8.6 extracted from the Project Evaluation Tool. Table 8.4 illustrates the working costs and royalties payable, Table 8.5 shows the capital expenditure, taxation and free cash flow calculations, and Table 8.6 shows the cash flow in nominal terms and the financial evaluation. The detailed working costs calculations are shown in Appendix 7.

The Project Evaluation Tool uses the discounted cash flow method in the determination of the viability of the project and the following outcomes/outputs resulted from this evaluation:

1. The net present value (NPV). This is the sum of the project cash flows over the period evaluated in today's money terms, discounted at a discount rate equal to the project's hurdle rate. The hurdle rate is a combination of the weighted average cost of capital (WACC) and an added risk premium to the study level, in this case a technical risk premium and the project phase risk premium. The latter was selected as the pre-feasibility from the corporate guidelines due to the additional optimization work required around the froth flotation circuit. The decision criteria for the project to be acceptable is that the NPV should be greater than 0 (zero).
2. The internal rate of return (IRR). This is equal to the hurdle rate if the NPV of the project is zero. If the NPV is greater than zero, the IRR will be greater than the hurdle rate. Therefore, the decision criteria for the IRR to be acceptable is if it is greater than the hurdle rate of the project.
3. The modified internal rate of return (MIRR). The IRR above assumes that the positive cash flows from a project are reinvested at the IRR. The IRR does not take into consideration the fact that the reinvestment rate may be lower than the IRR, therefore resulting in a bad project being considered. The MIRR assumes that all cash flows are reinvested at the company's WACC.

Therefore, MIRR more accurately reflects the profitability of the project. The decision criteria for an acceptable project would therefore be that the MIRR is greater than the project's hurdle rate.

4. Payback period. This is the period it will take before the funds invested is paid back by the project. The limitation for this measure is that it does not take into consideration the value of money over time. It, however, assists in determining which project has quick returns where projects with similar risk profiles and costs are compared. The decision criteria for the payback period to be acceptable is that it should have a maximum period of between 3 to 5 years depending on the project life.
5. Profitability index. This is a ratio of the present value of future cash flows to the initial cash outlay. The project is acceptable if it is greater than 1. If comparisons are done between several similar projects in terms of costs and risks, the projects will be preferentially ranked in order of highest to lowest.
6. Peak funding. This is an indication of how much cash outflow will the company spend before the project starts paying back returns. This is not a decision making criterion for accepting or rejecting projects. Rather, it requires proper cash flow planning to be performed for the project.
7. Discount rate. The discount rate is the percentage used to discount the cash flows of the project. It is derived from the WACC for the company plus the addition risk premiums such as project phase, country where the project is located as well as the associated level of technical risk of the project. This rate can only be reduced by the certainty of information at the evaluation date, thus the phase of the project and the technical risk reduction of the project are within certain limits as determined by the corporate finance department.
8. Nominal vs. Real. Real terms are those which one pays for an item assuming that there is no inflation effect on the item. The nominal price takes into consideration the effect of inflation upon the value of the money used.

Table 8.4: Post implementation financial analysis showing working costs and royalties payable (Powell and Botha, 2009)

	Unit	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Post-Implementation Revenue																		
Coking coal benchmark	Tonnes	-	-	-	-	-	-	-	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064
		-	-	-	-	-	-	-	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064
Coking coal benchmark	R	-	-	-	-	-	-	-	85 946 279	90 270 786	94 733 894	99 514 313	104 448 054	109 721 298	115 163 693	120 685 240	126 960 872	133 340 777
	R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Revenue	R	-	-	-	-	-	-	-	85 946 279	90 270 786	94 733 894	99 514 313	104 448 054	109 721 298	115 163 693	120 685 240	126 960 872	133 340 777
Operational cost																		
Manpower	Headcount	-	-	-	-	-	-	-	15	15	15	15	15	15	15	15	15	15
Manpower Nominal Costs	R/Headcount	-	-	-	-	-	-	17 424	18 469	19 577	20 752	21 997	23 317	24 716	26 199	27 771	29 437	31 203
Total Manpower Costs	R	-	-	-	-	-	-	-	277 037	293 659	311 279	329 955	349 753	370 738	392 982	416 561	441 555	468 048
Fixed Opex Cost	R	-	-	-	-	-	-	-	944 256	991 469	1 041 043	1 093 095	1 147 750	1 205 137	1 265 394	1 328 664	1 395 097	1 464 852
Total Fixed cost	R	-	-	-	-	-	-	-	1 372 952	1 513 680	1 668 832	1 839 887	2 028 476	2 236 395	2 465 625	2 718 352	2 996 983	3 304 173
Variable Opex Cost	R/t product	-	-	-	-	-	-	-	111	116	122	128	134	141	148	155	163	171
Total Variable cost	R	-	-	-	-	-	-	-	9 008 007	9 931 328	10 949 289	12 071 592	13 308 930	14 673 095	16 177 087	17 835 239	19 663 351	21 678 844
Electricity cost	R/t product	-	-	-	-	-	-	-	23	24	25	26	28	29	31	32	34	35
Total Electricity cost	R	-	-	-	-	-	-	-	2 818 560	3 107 463	3 425 978	3 777 140	4 164 297	4 591 138	5 061 729	5 580 557	6 152 564	6 783 201
Distribution cost	R/t product	-	-	-	-	-	-	-	90	97	103	108	114	119	125	131	138	145
Total Distribution cost	R	-	-	-	-	-	-	-	7 336 554	8 319 652	9 259 773	10 208 900	11 255 312	12 408 981	13 680 902	15 083 195	16 629 222	18 333 717
Total operational cost, excluding contingencies	R	-	-	-	-	-	-	-	20 813 111	23 165 782	25 615 151	28 227 474	31 106 767	34 280 346	37 778 326	41 633 902	45 883 673	50 567 984
Impact to Income																		
Incremental Revenue Generated	R	-	-	-	-	-	-	-	85 946 279	90 270 786	94 733 894	99 514 313	104 448 054	109 721 298	115 163 693	120 685 240	126 960 872	133 340 777
Operating Expenditure Changes, including contingency	R	-	-	-	-	-	-	-	24 975 733	27 798 939	30 738 181	33 872 969	37 328 121	41 136 416	45 333 991	49 960 683	55 060 408	60 681 581
Total	R	-	-	-	-	-	-	-	60 970 546	62 471 847	63 995 714	65 641 343	67 119 933	68 584 882	69 829 702	70 724 557	71 900 464	72 659 196
Working Capital Computation																		
Debtors	R	-	-	-	-	-	-	-	-7 064 078	-7 419 517	-7 786 347	-8 179 259	-8 584 772	-9 018 189	-9 465 509	-9 919 335	-10 435 140	-10 959 516
Trade creditors	R	-	-	-	-	-	-	-	2 052 800	2 284 844	2 526 426	2 784 080	3 068 065	3 381 075	3 726 081	4 106 357	4 525 513	4 987 527
Capital creditors	R	-	-	-	-	-	-	2 545 551	-	-	-	-	-	-	-	-	-	-
Net working capital	R	-	-	-	-	-	-	2 545 551	-5 011 278	-5 134 672	-5 259 922	-5 395 179	-5 516 707	-5 637 114	-5 739 428	-5 812 977	-5 909 627	-5 971 989
Movement in working capital	R	-	-	-	-	-	-	2 545 551	-7 556 829	-123 395	-125 249	-135 257	-121 528	-120 407	-102 314	-73 550	-96 650	-62 362
Royalties																		
Royalties																		
Total Revenue	R	-	-	-	-	-	-	-	85 946 279	90 270 786	94 733 894	99 514 313	104 448 054	109 721 298	115 163 693	120 685 240	126 960 872	133 340 777
Royalty Costs	R	-	-	-	-	-	-	-	859 463	902 708	947 339	995 143	1 044 481	1 097 213	1 151 637	1 206 852	1 269 609	1 333 408
Total Royalties	R	-	-	-	-	-	-	-	859 463	902 708	947 339	995 143	1 044 481	1 097 213	1 151 637	1 206 852	1 269 609	1 333 408

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Table 8.5: Post implementation financial analysis showing capital expenditure, taxation and free cash flow (Powell and Botha, 2009)

	Unit	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Capex Computation																		
Capex Amount	R	21 846 704																
Capital amount	R	18 205 587																
Capex Contingency @ 20%		3 641 117																
Capex Amount per period		-	-	-	-	-	-	21 846 704	-	-	-	-	-	-	-	-	-	-
Cumulative capital		-	-	-	-	-	-	21 846 704	21 846 704	21 846 704	21 846 704	21 846 704	21 846 704	21 846 704	21 846 704	21 846 704	21 846 704	21 846 704
Sustaining capital		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Applied R Inflation factor		1.00	1.09	1.17	1.22	1.29	1.35	1.42	1.49	1.56	1.64	1.72	1.81	1.90	1.99	2.09	2.20	2.31
Nominal Capex Amount per Timing		-	-	-	-	-	-	30 970 870	-	-	-	-	-	-	-	-	-	-
Total Nominal Capex	R	-	-	-	-	-	-	30 970 870	-	-	-	-	-	-	-	-	-	-
Wear and Tear % for an Option																		
Capex timing used		-	-	-	-	-	-	30 970 870	-	-	-	-	-	-	-	-	-	-
Total Wear & Tear Selected Option		-	-	-	-	-	-	30 970 870	-	-	-	-	-	-	-	-	-	-
Tax Computations																		
Net Profit	R	-	-	-	-	-	-	-	60 111 083	61 569 139	63 048 375	64 646 200	66 075 452	67 487 669	68 678 065	69 517 704	70 630 855	71 325 788
Less: Wear & Tear/Tax Allowance	R	-	-	-	-	-	-	-30 970 870	-	-	-	-	-	-	-	-	-	-
Taxable Income/Loss	R	-	-	-	-	-	-	-30 970 870	60 111 083	61 569 139	63 048 375	64 646 200	66 075 452	67 487 669	68 678 065	69 517 704	70 630 855	71 325 788
Tax Loss Control Account																		
Opening Balance	R	-	-	-	-	-	-	-	-30 970 870	29 140 213	90 709 352	153 757 727	218 403 927	284 479 380	351 967 049	420 645 114	490 162 819	560 793 674
Tax Loss for Period	R	-	-	-	-	-	-	-30 970 870	-	-	-	-	-	-	-	-	-	-
Taxable Income for the year	R	-	-	-	-	-	-	-	60 111 083	61 569 139	63 048 375	64 646 200	66 075 452	67 487 669	68 678 065	69 517 704	70 630 855	71 325 788
Closing Balance	R	-	-	-	-	-	-	-30 970 870	29 140 213	90 709 352	153 757 727	218 403 927	284 479 380	351 967 049	420 645 114	490 162 819	560 793 674	632 119 462
Taxable Income for the year		-	-	-	-	-	-	-	29 140 213	61 569 139	63 048 375	64 646 200	66 075 452	67 487 669	68 678 065	69 517 704	70 630 855	71 325 788
Tax Payable	R	-	-	-	-	-	-	-	8 159 260	17 239 359	17 653 545	18 100 936	18 501 127	18 896 547	19 229 858	19 464 957	19 776 639	19 971 221
Year									1	2	3	4	5	6	7	8	9	10
Free Cash Flow Computations																		
Operating Profit	R	-	-	-	-	-	-	-	60 111 083	61 569 139	63 048 375	64 646 200	66 075 452	67 487 669	68 678 065	69 517 704	70 630 855	71 325 788
Tax	R	-	-	-	-	-	-	-	-8 159 260	-17 239 359	-17 653 545	-18 100 936	-18 501 127	-18 896 547	-19 229 858	-19 464 957	-19 776 639	-19 971 221
Net Working Capital Movement	R	-	-	-	-	-	-	2 545 551	-7 556 829	-123 395	-125 249	-135 257	-121 528	-120 407	-102 314	-73 550	-96 650	-62 362
Capex	R	-	-	-	-	-	-	-30 970 870	-	-	-	-	-	-	-	-	-	-
Free Cash Flow	R	-	-	-	-	-	-	-28 425 319	44 394 995	44 206 386	45 269 581	46 410 007	47 452 798	48 470 715	49 345 893	49 979 198	50 757 566	51 292 206
Cumulative Free Cash Flow	R	-	-	-	-	-	-	-28 425 319	15 969 676	60 176 061	105 445 642	151 855 649	199 308 447	247 779 162	297 125 055	347 104 252	397 861 818	449 154 024

Table 8.6: Post implementation cash flow in nominal terms and financial evaluation (Powell and Botha, 2009)

Cash Flow in Nominal Terms						
Year	0	1	2	3	4	5
Financial Year	2015	2016	2017	2018	2019	2020
Revenue	-	85.09	89.37	93.79	98.52	103.40
Incremental Revenue Generated	-	85.95	90.27	94.73	99.51	104.45
Royalties	-	-0.86	-0.90	-0.95	-1.00	-1.04
Operating Expenditure Changes	-	-24.98	-27.80	-30.74	-33.87	-37.33
Capital	-30.97	-	-	-	-	-
Investment Expenditure	-24.78	-	-	-	-	-
Contingency	-6.19	-	-	-	-	-
Tax	-	-8.16	-17.24	-17.65	-18.10	-18.50
Working Capital Changes	2.55	-7.56	-0.12	-0.13	-0.14	-0.12
Cash Flow (after tax, WC & STC)	-28.43	44.39	44.21	45.27	46.41	47.45

Year	6	7	8	9	10
Financial Year	2021	2022	2023	2024	2025
Revenue	108.62	114.01	119.48	125.69	132.01
Incremental Revenue Generated	109.72	115.16	120.69	126.96	133.34
Royalties	-1.10	-1.15	-1.21	-1.27	-1.33
Operating Expenditure Changes	-41.14	-45.33	-49.96	-55.06	-60.68
Capital	-	-	-	-	-
Investment Expenditure	-	-	-	-	-
Contingency	-	-	-	-	-
Tax	-18.90	-19.23	-19.46	-19.78	-19.97
Working Capital	-0.12	-0.10	-0.07	-0.10	-0.06
Cash Flow (after tax, WC & STC)	48.47	49.35	49.98	50.76	51.29

Financial Evaluation Results				
Evaluation period -Year(s)	Ten	NPV @ Start of Jul-09 @ 23%	R'm	21.21
Total capital expenditure	R'm	IRR	%	153%
Corporate Tax Rate	%	MIRR		40%
Hurdle Rate	23.0%	Profitability Index		3.63
		Peak Funding	R'm	(28.43)
		Payback Period in Year(s)	years	7.64

Based on the results shown in Table 8.6 a positive NPV of R21.21 million as of July 2009 was calculated and the IRR, MIRR and a profitability index of 153%, 40% and 3.63 were achieved respectively. The IRR and, more importantly, MIRR were higher than the hurdle rate and indicate that the project is potentially viable. The peak funding would be experienced in 2015 ahead of the intended start of operations in 2016.

The project payback period from the model at 7.64 years in Table 8.6 is calculated from the date of the initiation of the project, and therefore the dormant years of the project, from 2009 to 2014 along with the actual expenditure in 2015 need to be subtracted from this value to get the true payback period. Therefore, Table 8.6 needs to be read in conjunction with both Table 8.4 and Table 8.5. The true payback period would therefore be 1.6 years which clearly show that the project is an attractive and very viable option since it is well below a Corporate Finance guideline of 3 to 7 years.

8.3.3.2 Revised Evaluation

The original study was revised in April 2015 to take into account economic changes that have occurred since the original work was completed in 2009. Only two changes were considered for this, namely the fall of the hard coking coal price and the Rand:US Dollar exchange rate. The Exxaro Project Evaluation Tool was again used as the platform for these calculations.

The Rand: US Dollar exchange rate on 14th April 2015 was R12.13 to \$1 (Bloomberg 2015). For the purposes of the model, an exchange rate of R12.00 to \$1 was used up to and including 2020, and thereafter the prevailing 2009 rates in the original macroeconomic model left as is. This was done to allow for a conservative future view.

The hard coking coal price has also seen to have dipped in recent years with the 2014 average for the first 3 quarters being \$99.10 per tonne (Steelonthenet 2015). The price used in the model was therefore \$100 per tonne for the entire period, which would provide a conservative view going forward.

The original and updated values are illustrated in Figure 8.5:

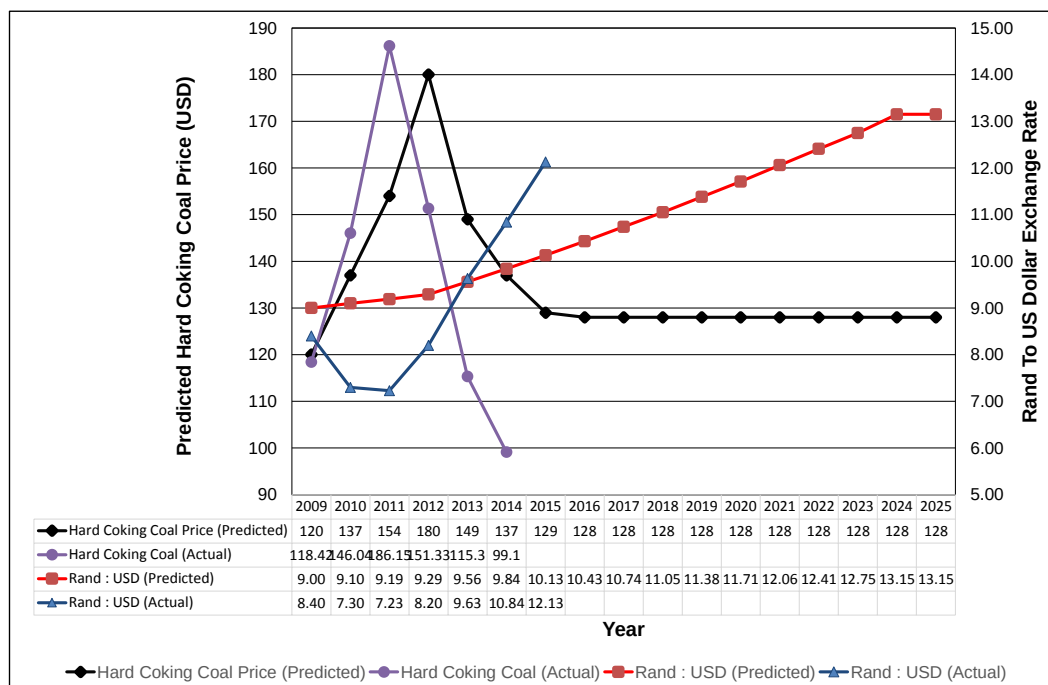


Figure 8.5: Comparison of the predicted and actual price of hard coking coal and the Rand:US Dollar exchanges rates 2009 to 2015

Using the updated data inputted into the Project Evaluation Tool for the April 2015 revision, the results are summarized in Table 8.7 for the working costs and royalties payable, Table 8.8 the capital expenditure, taxation and free cash flow calculations, and Table 8.9 the cash flow in nominal terms and the financial evaluation (which should be also be considered in relation to the other two tables).

Table 8.7: Post implementation financial analysis showing working costs and royalties payable 2015 update

	Unit	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Post-Implementation Revenue																		
Coking coal benchmark	Tonnes	-	-	-	-	-	-	-	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064
		-	-	-	-	-	-	-	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064
Coking coal benchmark	R	-	-	-	-	-	-	-	77 279 896	78 825 494	80 402 004	82 010 044	83 650 245	85 749 866	90 003 230	94 318 453	99 223 013	104 209 063
	R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Revenue	R	-	-	-	-	-	-	-	77 279 896	78 825 494	80 402 004	82 010 044	83 650 245	85 749 866	90 003 230	94 318 453	99 223 013	104 209 063
Operational cost																		
Manpower	Headcount	-	-	-	-	-	-	-	15	15	15	15	15	15	15	15	15	15
Manpower Nominal Costs	R/Headcount	-	-	-	-	-	-	17 424	18 469	19 577	20 752	21 997	23 317	24 716	26 199	27 771	29 437	31 203
Total Manpower Costs	R	-	-	-	-	-	-	-	277 037	293 659	311 279	329 955	349 753	370 738	392 982	416 561	441 555	468 048
Fixed Opex Cost	R	-	-	-	-	-	-	-	944 256	991 469	1 041 043	1 093 095	1 147 750	1 205 137	1 265 394	1 328 664	1 395 097	1 464 852
Total Fixed cost	R	-	-	-	-	-	-	-	1 372 952	1 513 680	1 668 832	1 839 887	2 028 476	2 236 395	2 465 625	2 718 352	2 996 983	3 304 173
Variable Opex Cost	R/t product	-	-	-	-	-	-	-	111	116	122	128	134	141	148	155	163	171
Total Variable cost	R	-	-	-	-	-	-	-	9 008 007	9 931 328	10 949 289	12 071 592	13 308 930	14 673 095	16 177 087	17 835 239	19 663 351	21 678 844
Electricity cost	R/t product	-	-	-	-	-	-	-	23	24	25	26	28	29	31	32	34	35
Total Electricity cost	R	-	-	-	-	-	-	-	2 818 560	3 107 463	3 425 978	3 777 140	4 164 297	4 591 138	5 061 729	5 580 557	6 152 564	6 783 201
Distribution cost	R/t product	-	-	-	-	-	-	-	90	97	103	108	114	119	125	131	138	145
Total Distribution cost	R	-	-	-	-	-	-	-	7 336 554	8 319 652	9 259 773	10 208 900	11 255 312	12 408 981	13 680 902	15 083 195	16 629 222	18 333 717
Total operational cost, excluding contingencies	R	-	-	-	-	-	-	-	20 813 111	23 165 782	25 615 151	28 227 474	31 106 767	34 280 346	37 778 326	41 633 902	45 883 673	50 567 984
Impact to Income																		
Incremental Revenue Generated	R	-	-	-	-	-	-	-	77 279 896	78 825 494	80 402 004	82 010 044	83 650 245	85 749 866	90 003 230	94 318 453	99 223 013	104 209 063
Operating Expenditure Changes, including contingency	R	-	-	-	-	-	-	-	24 975 733	27 798 939	30 738 181	33 872 969	37 328 121	41 136 416	45 333 991	49 960 683	55 060 408	60 681 581
Total	R	-	-	-	-	-	-	-	52 304 163	51 026 555	49 663 823	48 137 075	46 322 124	44 613 450	44 669 239	44 357 770	44 162 605	43 527 483
Working Capital Computation																		
Debtors	R	-	-	-	-	-	-	-	-6 351 772	-6 478 808	-6 608 384	-6 740 552	-6 875 363	-7 047 934	-7 397 526	-7 752 202	-8 155 316	-8 565 129
Trade creditors	R	-	-	-	-	-	-	-	2 052 800	2 284 844	2 526 426	2 784 080	3 068 065	3 381 075	3 726 081	4 106 357	4 525 513	4 987 527
Capital creditors	R	-	-	-	-	-	-	2 545 551	-	-	-	-	-	-	-	-	-	-
Net working capital	R	-	-	-	-	-	-	2 545 551	-4 298 972	-4 193 963	-4 081 958	-3 956 472	-3 807 298	-3 666 859	-3 671 444	-3 645 844	-3 629 803	-3 577 601
Movement in working capital	R	-	-	-	-	-	-	2 545 551	-6 844 523	105 009	112 005	125 486	149 174	140 439	-4 585	25 600	16 041	52 202
Royalties																		
Royalties																		
Total Revenue	R	-	-	-	-	-	-	-	77 279 896	78 825 494	80 402 004	82 010 044	83 650 245	85 749 866	90 003 230	94 318 453	99 223 013	104 209 063
Royalty Costs	R	-	-	-	-	-	-	-	772 799	788 255	804 020	820 100	836 502	857 499	900 032	943 185	992 230	1 042 091
Total Royalties	R	-	-	-	-	-	-	-	772 799	788 255	804 020	820 100	836 502	857 499	900 032	943 185	992 230	1 042 091

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Table 8.8: Post implementation financial analysis showing capital expenditure, taxation and free cash flow 2015 update

	Unit	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Capex Computation																		
Capex Amount	R	21 846 704																
Capital amount	R	18 205 587																
Capex Contingency @ 20%		3 641 117																
Capex Amount per period		-	-	-	-	-	-	21 846 704	-	-	-	-	-	-	-	-	-	-
Cumulative capital		-	-	-	-	-	-	21 846 704	21 846 704	21 846 704	21 846 704	21 846 704	21 846 704	21 846 704	21 846 704	21 846 704	21 846 704	21 846 704
Sustaining capital		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Applied R Inflation factor		1.00	1.09	1.17	1.22	1.29	1.35	1.42	1.49	1.56	1.64	1.72	1.81	1.90	1.99	2.09	2.20	2.31
Nominal Capex Amount per Timing		-	-	-	-	-	-	30 970 870	-	-	-	-	-	-	-	-	-	-
Total Nominal Capex	R	-	-	-	-	-	-	30 970 870	-	-	-	-	-	-	-	-	-	-
Wear and Tear % for an Option																		
Capex timing used		-	-	-	-	-	-	30 970 870	-	-	-	-	-	-	-	-	-	-
Total Wear & Tear Selected Option		-	-	-	-	-	-	30 970 870	-	-	-	-	-	-	-	-	-	-
Tax Computations																		
Net Profit	R	-	-	-	-	-	-	-	51 531 364	50 238 300	48 859 803	47 316 974	45 485 622	43 755 951	43 769 207	43 414 586	43 170 375	42 485 392
Less: Wear & Tear/Tax Allowance	R	-	-	-	-	-	-	-30 970 870	-	-	-	-	-	-	-	-	-	-
Taxable Income/Loss	R	-	-	-	-	-	-	-30 970 870	51 531 364	50 238 300	48 859 803	47 316 974	45 485 622	43 755 951	43 769 207	43 414 586	43 170 375	42 485 392
Tax Loss Control Account																		
Opening Balance	R	-	-	-	-	-	-	-	-30 970 870	20 560 494	70 798 794	119 658 597	166 975 571	212 461 193	256 217 144	299 986 351	343 400 937	386 571 312
Tax Loss for Period	R	-	-	-	-	-	-	-30 970 870	-	-	-	-	-	-	-	-	-	-
Taxable Income for the year	R	-	-	-	-	-	-	-	51 531 364	50 238 300	48 859 803	47 316 974	45 485 622	43 755 951	43 769 207	43 414 586	43 170 375	42 485 392
Closing Balance	R	-	-	-	-	-	-	-30 970 870	20 560 494	70 798 794	119 658 597	166 975 571	212 461 193	256 217 144	299 986 351	343 400 937	386 571 312	429 056 704
Taxable Income for the year		-	-	-	-	-	-	-	20 560 494	50 238 300	48 859 803	47 316 974	45 485 622	43 755 951	43 769 207	43 414 586	43 170 375	42 485 392
Tax Payable	R	-	-	-	-	-	-	-	5 756 938	14 066 724	13 680 745	13 248 753	12 735 974	12 251 666	12 255 378	12 156 084	12 087 705	11 895 910
Year									1	2	3	4	5	6	7	8	9	10
Free Cash Flow Computations																		
Operating Profit	R	-	-	-	-	-	-	-	51 531 364	50 238 300	48 859 803	47 316 974	45 485 622	43 755 951	43 769 207	43 414 586	43 170 375	42 485 392
Tax	R	-	-	-	-	-	-	-	-5 756 938	-14 066 724	-13 680 745	-13 248 753	-12 735 974	-12 251 666	-12 255 378	-12 156 084	-12 087 705	-11 895 910
Net Working Capital Movement	R	-	-	-	-	-	-	2 545 551	-6 844 523	105 009	112 005	125 486	149 174	140 439	-4 585	25 600	16 041	52 202
Capex	R	-	-	-	-	-	-	-30 970 870	-	-	-	-	-	-	-	-	-	-
Free Cash Flow	R	-	-	-	-	-	-	-28 425 319	38 929 902	36 276 585	35 291 063	34 193 708	32 898 822	31 644 724	31 509 244	31 284 102	31 098 711	30 641 684
Cumulative Free Cash Flow	R	-	-	-	-	-	-	-28 425 319	10 504 583	46 781 168	82 072 232	116 265 940	149 164 761	180 809 485	212 318 729	243 602 831	274 701 541	305 343 225

Table 8.9: Post implementation cash flow in nominal terms and financial evaluation 2015 update

Cash Flow in Nominal Terms						
Year	0	1	2	3	4	5
Financial Year	2015	2016	2017	2018	2019	2020
Revenue	-	76.51	78.04	79.60	81.19	82.81
Incremental Revenue Generated	-	77.28	78.83	80.40	82.01	83.65
Royalties	-	-0.77	-0.79	-0.80	-0.82	-0.84
Operating Expenditure Changes	-	-24.98	-27.80	-30.74	-33.87	-37.33
Capital	-30.97	-	-	-	-	-
Investment Expenditure	-24.78	-	-	-	-	-
Contingency	-6.19	-	-	-	-	-
Tax	-	-5.76	-14.07	-13.68	-13.25	-12.74
Working Capital Changes	2.55	-6.84	0.11	0.11	0.13	0.15
Cash Flow (after tax, WC & STC)	-28.43	38.93	36.28	35.29	34.19	32.90

Year	6	7	8	9	10
Financial Year	2021	2022	2023	2024	2025
Revenue	84.89	89.10	93.38	98.23	103.17
Incremental Revenue Generated	85.75	90.00	94.32	99.22	104.21
Royalties	-0.86	-0.90	-0.94	-0.99	-1.04
Operating Expenditure Changes	-41.14	-45.33	-49.96	-55.06	-60.68
Capital	-	-	-	-	-
Investment Expenditure	-	-	-	-	-
Contingency	-	-	-	-	-
Tax	-12.25	-12.26	-12.16	-12.09	-11.90
Working Capital	0.14	-0.00	0.03	0.02	0.05
Cash Flow (after tax, WC & STC)	31.64	31.51	31.28	31.10	30.64

Financial Evaluation Results				
Evaluation period -Year(s)	Ten	NPV @ Start of Jul-09 @ 23%	R'm	15.90
Total capital expenditure	R'm	IRR	%	127%
Corporate Tax Rate	%	MIRR		37%
Hurdle Rate	23.0%	Profitability Index		2.97
		Peak Funding	R'm	(28.43)
		Payback Period in Year(s)	years	7.73

Based on the results shown in Table 8.9, a positive NPV of R15.90 million was calculated which was some R5.3 million lower than the original evaluation performed in 2009. The difference being the changes to the Rand:US Dollar exchange rates and expected hard coking coal prices as described earlier.

The IRR and MIRR exceed the 23% hurdle rate and returned values of 127% and 37% respectively, and as expected lower than the original values by 16% in terms of the IRR and 3% for the MIRR. The project is therefore still viable according to these results.

The peak funding did not change since it would be incurred in 2015 as in the original evaluation.

The profitability index was determined to be 2.97 compared to the original value of 3.63.

The project payback period increased marginally to 7.73 compared to the original value of at 7.64 years. In operational terms, the payback period would be 1.73 years, and again remains a very positive indication of the viability of the project.

However, it should be borne in mind that increases for electricity were not considered and annual increases of 16% have been requested by Eskom (the state own power utility) in 2013 (Polity 2013) for the years 2013 to 2018 inclusive, though the National Energy Regulator of South Africa (Nersa) only approved an 8% increase. Despite this, NERSA approved an increase of 12.69% applicable from 1st April 2015 for direct users such as Tshikondeni (Eskom 2015).

Likewise, the distribution costs for moving the product from the mine to the siding at Musina, are probably higher than that portrayed in the calculations due to the full effect of increases in diesel required. The diesel price from 1st December 2009 to 1st April 2015, as reported by the Automobile Association of South Africa is shown in Table 8.10 below

Table 8.10: Annual variation in diesel price from 2009 to 2015

Year	Diesel Price (R/litre)
1st December 2009	7.04
1st December 2010	7.61
1st December 2011	10.49
1st December 2012	11.39
1st December 2013	12.55
1st December 2014	11.32
1st April 2015	11.23

It is seen that the pricing is variable, but the overall trend is upwards, but future predictions are very difficult to make, especially since this price is linked to the Rand:US Dollar exchange rate.

Despite the difficulties in future forecasting due to global market volatilities of the hard coking coal pricing, and uncertainties in respect to future electricity and fuel costs, the project would potentially be viable. Unfortunately the project was ultimately not pursued further by Exxaro since it did not fit into their current strategy,

nor that of the financially troubled ArcelorMittal SA who would essentially be required to fund the project since they own the mine with Exxaro operating it for them.

8.4 Simplification of the Process Flowsheet

Given that the above approach did work but gave a low yield of 10%, but required the modification of the plant to include milling, filtration and briquetting, the obvious next step would be: to consider whether or not the flowsheet could be simplified yet return a similar yield? From the information presented in Section 4.4.3.1, which considered only destoning using only screening and dense medium separation, it would appear that this would be possible with a yield ranging between 6% to 14% being achieved. For completeness of the work, the above financial evaluation process was also applied to this scenario i.e. removal of milling, flotation, filtration and briquetting, using the 2015 macroeconomic assumptions for a yield of 10% (the average of the results from Section 4.4.3.1). The estimated electricity usage was reduced from 150 to 75 to allow for removal of equipment previously mentioned. An average yield of 10% as suggested by the aforementioned work was also used. The estimated capital costs would therefore be significantly reduced from R21 846 704 to R1 481 430 as detailed in Table 8.11, which are the 2009 estimates prior to escalation.

Table 8.11: Estimated capital expenditure to modify the Tshikondeni plant for the retreatment of discards

Capital Item	Cost (R)
Screen	231 525
Sub Total 1 (Equipment)	231 525
Civil and Construction	1 000 000
Sub Total 2 (Equipment, Civils)	1 231 525
Contingency (20% of Sub Total 2)	243 305
Total	1 481 430

The results from the Project Evaluation Tool are summarized in Table 8.12 for the working costs and royalties payable, Table 8.13 the capital expenditure, taxation and free cash flow calculations, and Table 8.14 the cash flow in nominal terms and the financial evaluation (which should be read in conjunction with the two previous tables).

Table 8.12: Post implementation financial analysis showing working costs and royalties payable

	Unit	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Post-Implementation Revenue																		
Coking coal benchmark	Tonnes	-	-	-	-	-	-	-	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064
		-	-	-	-	-	-	-	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064
Coking coal benchmark	R	-	-	-	-	-	-	-	77 279 896	78 825 494	80 402 004	82 010 044	83 650 245	85 749 866	90 003 230	94 318 453	99 223 013	104 209 063
	R	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Revenue	R	-	-	-	-	-	-	-	77 279 896	78 825 494	80 402 004	82 010 044	83 650 245	85 749 866	90 003 230	94 318 453	99 223 013	104 209 063
Operational cost																		
Manpower	Headcount	-	-	-	-	-	-	-	15	15	15	15	15	15	15	15	15	15
Manpower Nominal Costs	R/Headcount	-	-	-	-	-	-	17 424	18 469	19 577	20 752	21 997	23 317	24 716	26 199	27 771	29 437	31 203
Total Manpower Costs	R	-	-	-	-	-	-	-	277 037	293 659	311 279	329 955	349 753	370 738	392 982	416 561	441 555	468 048
Fixed Opex Cost	R	-	-	-	-	-	-	-	944 256	991 469	1 041 043	1 093 095	1 147 750	1 205 137	1 265 394	1 328 664	1 395 097	1 464 852
Total Fixed cost	R	-	-	-	-	-	-	-	1 372 952	1 513 680	1 668 832	1 839 887	2 028 476	2 236 395	2 465 625	2 718 352	2 996 983	3 304 173
Variable Opex Cost	R/t product	-	-	-	-	-	-	-	111	116	122	128	134	141	148	155	163	171
Total Variable cost	R	-	-	-	-	-	-	-	9 008 007	9 931 328	10 949 289	12 071 592	13 308 930	14 673 095	16 177 087	17 835 239	19 663 351	21 678 844
Electricity cost	R/t product	-	-	-	-	-	-	-	23	24	25	26	28	29	31	32	34	35
Total Electricity cost	R	-	-	-	-	-	-	-	2 818 560	3 107 463	3 425 978	3 777 140	4 164 297	4 591 138	5 061 729	5 580 557	6 152 564	6 783 201
Distribution cost	R/t product	-	-	-	-	-	-	-	90	97	103	108	114	119	125	131	138	145
Total Distribution cost	R	-	-	-	-	-	-	-	7 336 554	8 319 652	9 259 773	10 208 900	11 255 312	12 408 981	13 680 902	15 083 195	16 629 222	18 333 717
Total operational cost, excluding contingencies	R	-	-	-	-	-	-	-	19 403 831	21 612 051	23 902 162	26 338 904	29 024 619	31 984 778	35 247 461	38 843 624	42 807 392	47 176 383
Impact to Income																		
Incremental Revenue Generated	R	-	-	-	-	-	-	-	77 279 896	78 825 494	80 402 004	82 010 044	83 650 245	85 749 866	90 003 230	94 318 453	99 223 013	104 209 063
Operating Expenditure Changes, including contingency	R	-	-	-	-	-	-	-	23 284 597	25 934 461	28 682 594	31 606 685	34 829 542	38 381 733	42 296 953	46 612 349	51 368 870	56 611 660
Total	R	-	-	-	-	-	-	-	53 995 299	52 891 033	51 719 410	50 403 359	48 820 702	47 368 133	47 706 277	47 706 104	47 854 143	47 597 404
Working Capital Computation																		
Debtors	R	-	-	-	-	-	-	-	-6 351 772	-6 478 808	-6 608 384	-6 740 552	-6 875 363	-7 047 934	-7 397 526	-7 752 202	-8 155 316	-8 565 129
Trade creditors	R	-	-	-	-	-	-	-	1 913 802	2 131 600	2 357 473	2 597 810	2 862 702	3 154 663	3 476 462	3 831 152	4 222 099	4 653 013
Capital creditors	R	-	-	-	-	-	-	172 195	-	-	-	-	-	-	-	-	-	-
Net working capital	R	-	-	-	-	-	-	172 195	-4 437 970	-4 347 208	-4 250 910	-4 142 742	-4 012 660	-3 893 271	-3 921 064	-3 921 050	-3 933 217	-3 912 115
Movement in working capital	R	-	-	-	-	-	-	172 195	-4 610 165	90 762	96 298	108 169	130 081	119 389	-27 793	14	-12 168	21 102
Royalties																		
Royalties																		
Total Revenue	R	-	-	-	-	-	-	-	77 279 896	78 825 494	80 402 004	82 010 044	83 650 245	85 749 866	90 003 230	94 318 453	99 223 013	104 209 063
Royalty Costs	R	-	-	-	-	-	-	-	772 799	788 255	804 020	820 100	836 502	857 499	900 032	943 185	992 230	1 042 091
Total Royalties	R	-	-	-	-	-	-	-	772 799	788 255	804 020	820 100	836 502	857 499	900 032	943 185	992 230	1 042 091

Chapter 8: Proposed Plant Design and Economic Evaluation

Table 8.13: Post implementation financial analysis showing capital expenditure, taxation and free cash flow

	Unit	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Capex Computation																		
Capex Amount	R	1 477 830																
Capital amount	R	1 231 525																
Capex Contingency @ 20%		246 305																
Capex Amount per period		-	-	-	-	-	-	1 477 830	-	-	-	-	-	-	-	-	-	-
Cumulative capital		-	-	-	-	-	-	1 477 830	1 477 830	1 477 830	1 477 830	1 477 830	1 477 830	1 477 830	1 477 830	1 477 830	1 477 830	1 477 830
Sustaining capital		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Applied R Inflation factor		1.00	1.09	1.17	1.22	1.29	1.35	1.42	1.49	1.56	1.64	1.72	1.81	1.90	1.99	2.09	2.20	2.31
Nominal Capex Amount per Timing		-	-	-	-	-	-	1 477 830	-	-	-	-	-	-	-	-	-	-
Total Nominal Capex	R	-	-	-	-	-	-	1 477 830	-	-	-	-	-	-	-	-	-	-
Wear and Tear % for an Option																		
Capex timing used		-	-	-	-	-	-	1 477 830	-	-	-	-	-	-	-	-	-	-
Total Wear & Tear Selected Option		-	-	-	-	-	-	1 477 830	-	-	-	-	-	-	-	-	-	-
Tax Computations																		
Net Profit	R	-	-	-	-	-	-	-	53 222 500	52 102 778	50 915 389	49 583 258	47 984 200	46 510 634	46 806 244	46 762 920	46 861 913	46 555 313
Less: Wear & Tear/Tax Allowance	R	-	-	-	-	-	-	-2 095 038	-	-	-	-	-	-	-	-	-	-
Taxable Income/Loss	R	-	-	-	-	-	-	-2 095 038	53 222 500	52 102 778	50 915 389	49 583 258	47 984 200	46 510 634	46 806 244	46 762 920	46 861 913	46 555 313
Tax Loss Control Account																		
Opening Balance	R	-	-	-	-	-	-	-	-2 095 038	51 127 462	103 230 240	154 145 629	203 728 888	251 713 088	298 223 722	345 029 966	391 792 886	438 654 798
Tax Loss for Period	R	-	-	-	-	-	-	-2 095 038	-	-	-	-	-	-	-	-	-	-
Taxable Income for the year	R	-	-	-	-	-	-	-	53 222 500	52 102 778	50 915 389	49 583 258	47 984 200	46 510 634	46 806 244	46 762 920	46 861 913	46 555 313
Closing Balance	R	-	-	-	-	-	-	-2 095 038	51 127 462	103 230 240	154 145 629	203 728 888	251 713 088	298 223 722	345 029 966	391 792 886	438 654 798	485 210 111
Taxable Income for the year		-	-	-	-	-	-	-	51 127 462	52 102 778	50 915 389	49 583 258	47 984 200	46 510 634	46 806 244	46 762 920	46 861 913	46 555 313
Tax Payable	R	-	-	-	-	-	-	-	14 315 689	14 588 778	14 256 309	13 883 312	13 435 576	13 022 978	13 105 748	13 093 618	13 121 336	13 035 488
Year									1	2	3	4	5	6	7	8	9	10
Free Cash Flow Computations																		
Operating Profit	R	-	-	-	-	-	-	-	53 222 500	52 102 778	50 915 389	49 583 258	47 984 200	46 510 634	46 806 244	46 762 920	46 861 913	46 555 313
Tax	R	-	-	-	-	-	-	-	-14 315 689	-14 588 778	-14 256 309	-13 883 312	-13 435 576	-13 022 978	-13 105 748	-13 093 618	-13 121 336	-13 035 488
Net Working Capital Movement	R	-	-	-	-	-	-	172 195	-4 610 165	90 762	96 298	108 169	130 081	119 389	-27 793	14	-12 168	21 102
Capex	R	-	-	-	-	-	-	-2 095 038	-	-	-	-	-	-	-	-	-	-
Free Cash Flow	R	-	-	-	-	-	-	-1 922 843	34 296 646	37 604 762	36 755 378	35 808 115	34 678 705	33 607 046	33 672 703	33 669 316	33 728 410	33 540 927
Cumulative Free Cash Flow	R	-	-	-	-	-	-	-1 922 843	32 373 803	69 978 564	106 733 943	142 542 057	177 220 763	210 827 808	244 500 512	278 169 828	311 898 238	345 439 165

Table 8.14: Post implementation cash flow in nominal terms and financial evaluation

Cash Flow in Nominal Terms						
Year	0	1	2	3	4	5
Financial Year	2015	2016	2017	2018	2019	2020
Revenue	-	77.28	78.83	80.40	82.01	82.81
Incremental Revenue Generated	-	-0.77	-0.79	-0.80	-0.82	83.65
Royalties	-	-23.28	-25.93	-28.68	-31.61	-0.84
Operating Expenditure Changes	-2.10	-	-	-	-	-34.83
Capital	-1.68	-	-	-	-	-
Investment Expenditure	-0.42	-	-	-	-	-
Contingency	-	-14.32	-14.59	-14.26	-13.88	-
Tax	0.17	-4.61	0.09	0.10	0.11	-13.44
Working Capital Changes	-1.92	34.30	37.60	36.76	35.81	0.13
Cash Flow (after tax, WC & STC)	-28.43	38.93	36.28	35.29	34.19	34.68

Year	6	7	8	9	10
Financial Year	2021	2022	2023	2024	2025
Revenue	84.89	89.10	93.38	98.23	103.17
Incremental Revenue Generated	85.75	90.00	94.32	99.22	104.21
Royalties	-0.86	-0.90	-0.94	-0.99	-1.04
Operating Expenditure Changes	-38.38	-42.30	-46.61	-51.37	-56.61
Capital	-	-	-	-	-
Investment Expenditure	-	-	-	-	-
Contingency	-	-	-	-	-
Tax	-13.02	-13.11	-13.09	-13.12	-13.04
Working Capital	0.12	-0.03	0.00	-0.01	0.02
Cash Flow (after tax, WC & STC)	33.61	33.67	33.67	33.73	33.54

Financial Evaluation Results				
Evaluation period -Year(s)	Ten	NPV @ Start of Jul-09 @ 23%	R'm	22.44
Total capital expenditure	2.10	IRR	%	1793%
Corporate Tax Rate	28.0%	MIRR		79%
Hurdle Rate	23.0%	Profitability Index		42.13
		Peak Funding	R'm	(1.92)
		Payback Period in Year(s)	years	7.06

Even after reducing the capital expenditure significantly to an escalated cost of approximately R1.92 million, a positive NPV of R22.44 million was determined, an incredibly high IRR of 1793% with the MIRR showing 79%. The profitability index was also high at 42.13. The project payback period was 7.06 years which after subtraction of the lack of activity between 2009 to 2014, and only capital expenses incurred in 2015, suggests that the project payback is actually 1.06 years. Given that these changes to the plant would be readily made and limited civils and construction work being required, the plant could probably be re-commissioned during 2015 and thus payback in an even shorter time.

The flotation section could also be re-commissioned to improve the overall yield achievable, though extensive optimization would be required to successfully process the slow floating fines.

8.5 Chapter Summary

A detailed financial evaluation of the project was undertaken using the Project Evaluation Tool developed by Exxaro's Corporate Finance Department. The original work from 2009 was presented along with an update for 2015.

The capital requirement for the plant was kept at a relatively low value since the project could be undertaken by modification of the plant after the ROM sources had been depleted.

In both versions, the project exceeded the hurdle rates as per requirements, and the payback period in real terms would be just under 2 years for both the original 2009 and updated 2015 evaluations. This makes the project very viable for Exxaro, but unfortunately the project was not pursued further since it was not in line with the strategic plans for either Exxaro or ArcelorMittal SA.

Simplification of the plant has also shown that the plant at 10% yield from a screening and dense medium process was also viable and paid back in marginally over 1 year. The flotation circuit would need some optimization to process the fines and could add additional tonnes to the final product.

The unknown factors such as future Rand:US Dollar exchange rate, the hard coking coal price, and fuel and electricity costs though should not impact upon the project too severely.

8.6 References

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Chapter 9: Preventing the Losses by Reconfiguring the Plant for Fine Coal Dense Medium Separation

Based on the work conducted on the various discard samples considered within this work it is obvious that saleable product, especially within the finer sizes, is present. The economic recovery of these losses was found to be possible albeit at a low yield (10%) and short payback period of less than 2 years. The discard retreatment plant assumes that the plant is modified to process discards after the exhaustion of the run of mine sources, which would result in the losses to the discards streams, and the failure to prevent these losses by optimization of the existing plant.

This then leads to the question: is prevention better than cure? In other words, should consideration have been given to modification of the plant to proactively prevent these losses? The answer is yes, it was. Unfortunately, just after completion of this section of the work, the world went into a major global depression which seriously affected the sole client of Tshikondeni Coal, namely ArcelorMittal South Africa, who were affected seriously during this period. However, this body of work would be incomplete without it being considered since commodity prices are cyclical and would eventually rise. This section of the work is largely rhetorical/philosophical given that the mine entered its closure phase in early 2015 as per the operational plan.

9.1 Reconfiguration of the Fines Circuit to Include Fine Coal Dense Medium Separation

9.1.1 Introduction

Previous internal yield optimization projects performed in conjunction with Exxaro's Research and Development department had indicated that flotation was not the most efficient process for the +0.5mm fraction, and that the application of either a teetered bed separator or a dense medium cyclone would produce better results, particularly in the case of the material received from Mutale Shaft which did not respond well to this technique (Rabe, 2004; Van Wyngaard, 2005; Van der Merwe, 2006; Van der Merwe, 2007). This was also confirmed at the start of this work as detailed in Section 2.4.

Prior to January 2004, the flotation tailings were treated in a spiral plant to recover losses of saleable product within this size fraction. However, the spirals product was typically in the order of 17% ash and the final overall product specification could be achieved by operating the dense medium cyclones processing the -13+1.4mm material at lower densities. The consequence of this was a lowering of the plant yield, and ultimately the spiral plant was decommissioned in January 2004 (Pretorius, 2007).

During 2005, a bulk sample of Tshikondeni run of mine material was transported to and tested through a fines dense medium cyclone demonstration plant located at Anglo American's Navigation Colliery as part of the coal processing component of the South African Coaltech programme, an initiative between industry and academia for all aspects of the coal mining industry, and managed through the Council for Scientific and Industrial Research (the CSIR). The report by De Korte and McGonigal (2005) clearly indicated that saleable product could be produced by this technique and as seen in Figure 9.1, a 13.7% ash product was made at a yield of 74.4% and an acceptable E_p of 0.0537, which is better than would be expected from a spiral, and probably a flotation plant (though this is not easy to determine due to

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the fundamental differences in the respective separation processes viz. gravity versus surface chemistry).

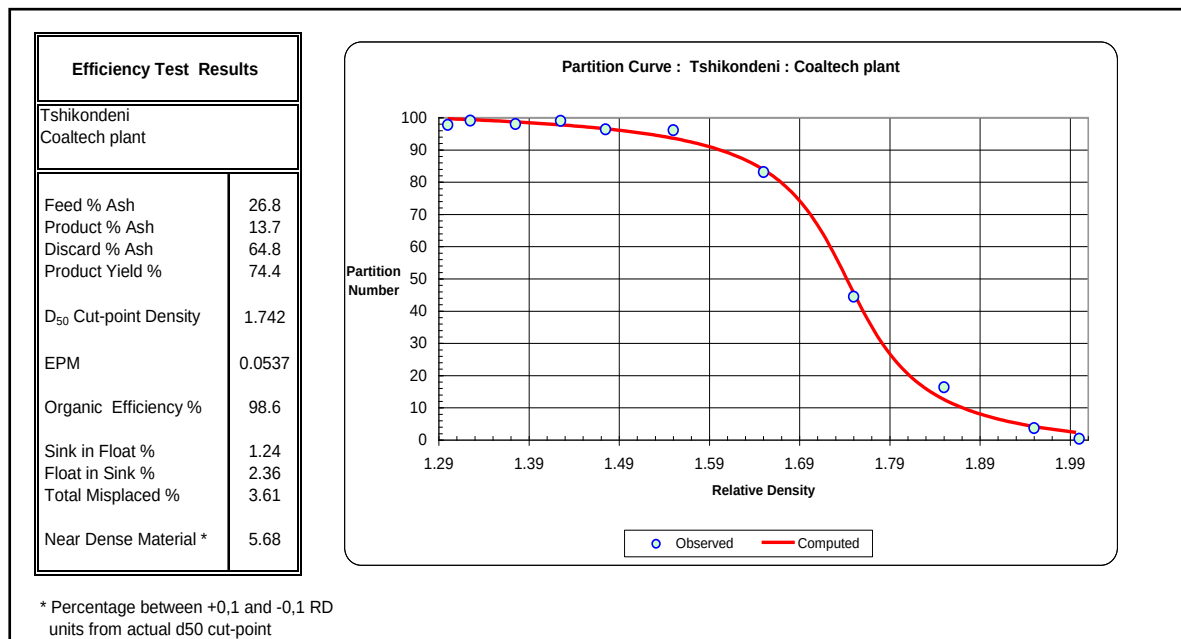


Figure 9.1: Summary of pilot test in Tshikondeni run of mine tested through the Coaltech fines dense medium cyclone plant (De Korte and McGonigal, 2005)

This clearly illustrated that the potential to apply this technique at Tshikondeni existed and should be explored in more detail. Further work was therefore undertaken to consider the reconfiguration of the fines circuit within the Tshikondeni plant to allow for processing of the fines fraction via dense medium separation. A total of five separation circuits were considered:

- Scenario 1: All to 710 cyclone
- Scenario 2: -1.4mm to new 420 cyclone; +1.4mm to current 710 cyclone
- Scenario 3: -0.212mm to flotation; +0.212mm to 710 cyclones
- Scenario 4: -0.212 to flotation; -1.4+0.212 to 420 cyclone; +1.4mm to current 710 cyclone
- Scenario 5: -3mm to new 420 cyclone; +3mm to current 710 cyclone

The various scenarios are discussed below in greater detail.

9.1.2 Scenario 1

The first scenario considered the concept of “wash to zero”, well described by De Korte (2000), whereby the whole of the plant feed is processed through a dense medium cyclone without any desliming taking place. The simulation indicated that the current 710mm cyclones will have enough capacity. The cyclone product and discard go to the current drain and rinse screens where the -1mm is diverted to new magnetic separators. The discard magnetic separator underflow is pumped to the current flotation tailsings system, and the product magnetic separator underflow will go to the centrifuge. The spiral feed dewatering screen can also be used if necessary. The plant feed rate is taken as 180 t/hour (90t/hour per module). The process flow diagram is illustrated in Figure 9.2:

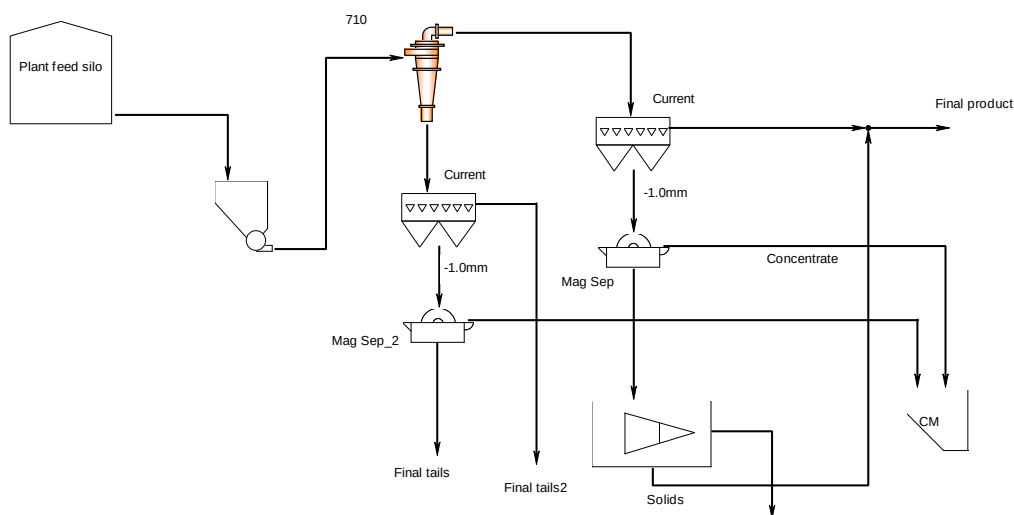


Figure 9.2: Scenario 1 process flow diagram

The main advantage of this approach is a simplified plant circuit, and hence capital and operating costs. Plant efficiency above 0.5mm is typically comparable with standard circuits, though loss of efficiency is experienced with the -0.5mm material and expected E_p values in the order of 0.150 to 0.200 would be similar to or marginally better than a spiral. This negates the need for the installation of a spiral circuit within the plant flowsheet, though the -0.5mm may create problems with overall density control due to this size fraction contributing to medium viscosity. Furthermore, should the washability properties of the feed change significantly,

product quality issues may arise from the inability to control the operating density for the -1.4mm material.

The other risk is the capacity of the drain and rinse screens being constrained, meaning the plant feed rate would need to be reduced from 180t/hour.

9.1.3 Scenario 2

For the Tshikondeni material, as seen in Table 0.5 later in the chapter, this was the second least favourable option, though the largest operational saving could be realized. This configuration adds a dense medium cyclone circuit for the -1.4mm fraction in place of the current flotation. This circuit will be a replica of Exxaro's Leeuwpan coal mine circuit. The discard magnetic separator underflow will go to the flotation tailings system and the product magnetic separator underflow will go the centrifuge. The spiral feed dewatering screens can also be used if necessary. The -1.4mm fraction will be 54 t/hour and will consist of two 27 ton/hour modules to correspond with the plant feed rate of 90t/hour per module. The process flow diagram is depicted in Figure 9.3.

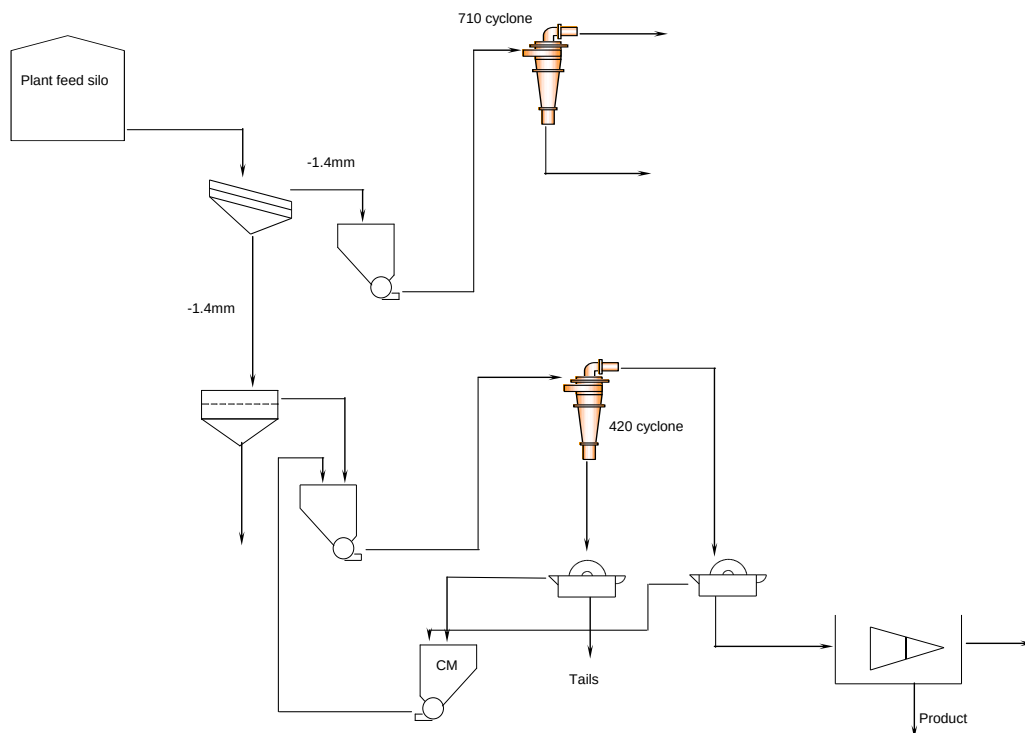


Figure 9.3: Scenario 2 process flow diagram

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Pretorius (2007) reported that the plant was tested in a manner similar to this scenario, except that -0.5mm material was used as froth flotation feed. The flotation circuit was noted to show an improvement, though overall yield was lost due to poor response of the -1.4+0.5mm material in the cyclone, and concurred with the anticipated risk of processing the -1.4+0.212mm material in the cyclone.

9.1.5 Scenario 4

Scenario 4 is almost the same as scenario 2, with the exception that the -0.212mm fraction is beneficiated by flotation. This is done to increase the efficiency of the flotation as well as the dense medium cyclones. Previous test work indicated that flotation is the best method to beneficiate the -0.5mm fraction. (Van der Merwe 2006, Van Wyngaard 2004), allied with the efficiency for dense medium beneficiation decreasing exponentially as the particle size decreases (De Korte 2000). The process flow diagram is illustrated in Figure 9.5.

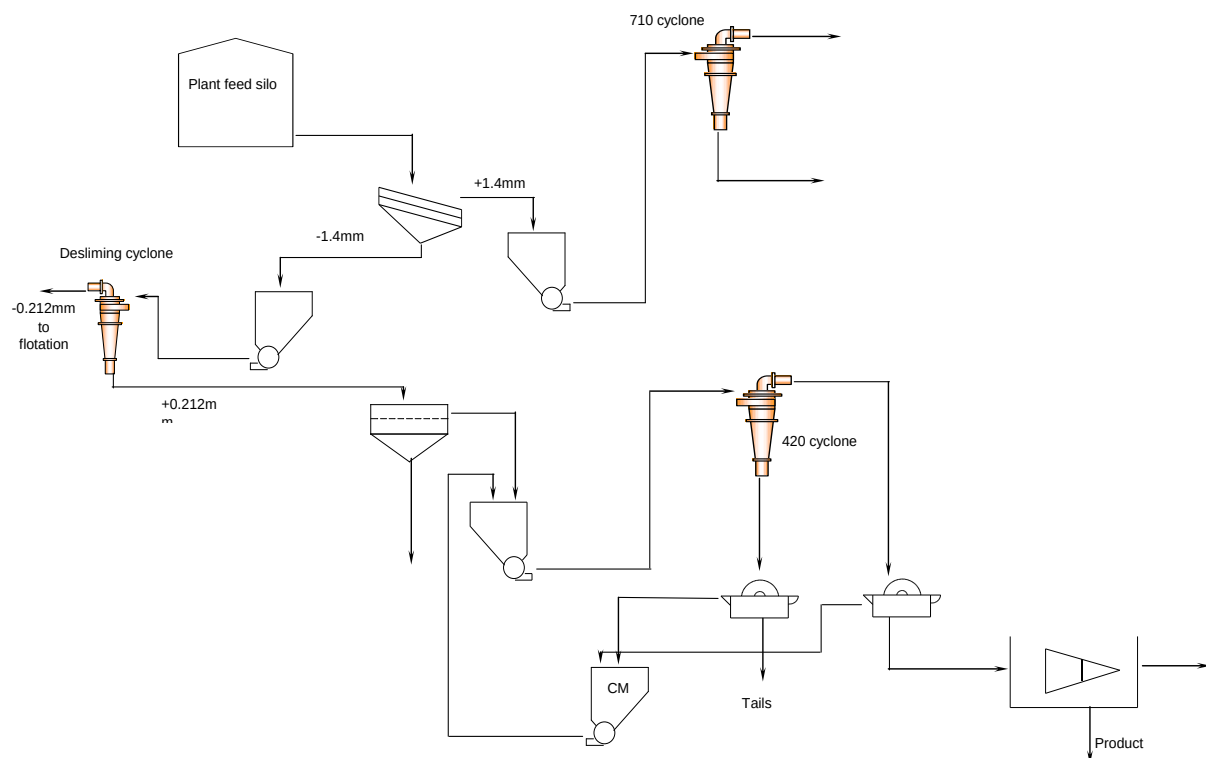


Figure 9.5: Scenario 4 process flow diagram

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The -1.4mm will be deslimed with a cyclone. The +0.212mm will be dewatered on a screen before it is beneficiated in a 420mm cyclone. The -0.212mm will go the current flotation circuit. The fine coal dense medium circuit will be a replica of the Leeuwpan circuit. The flotation discard and discard magnetic separator underflow is handled together, and the flotation product is added with the product magnetic separator underflow to the centrifuge. The -0.212mm will be around 18t/hour with the -1.4+0.212mm fraction at 36t/hour. A single unit for the latter would be sufficient, but may not fit in with the plant operating philosophy should a breakdown occur in this section, i.e. the whole plant would stand during this stoppage.

9.1.6 Scenario 5

This alternative adds a dense medium cyclone circuit for the -3mm fraction in place of the current flotation. The plant feed screen will be changed to 3mm from the current 1.4mm. It is illustrated in Figure 9.6.

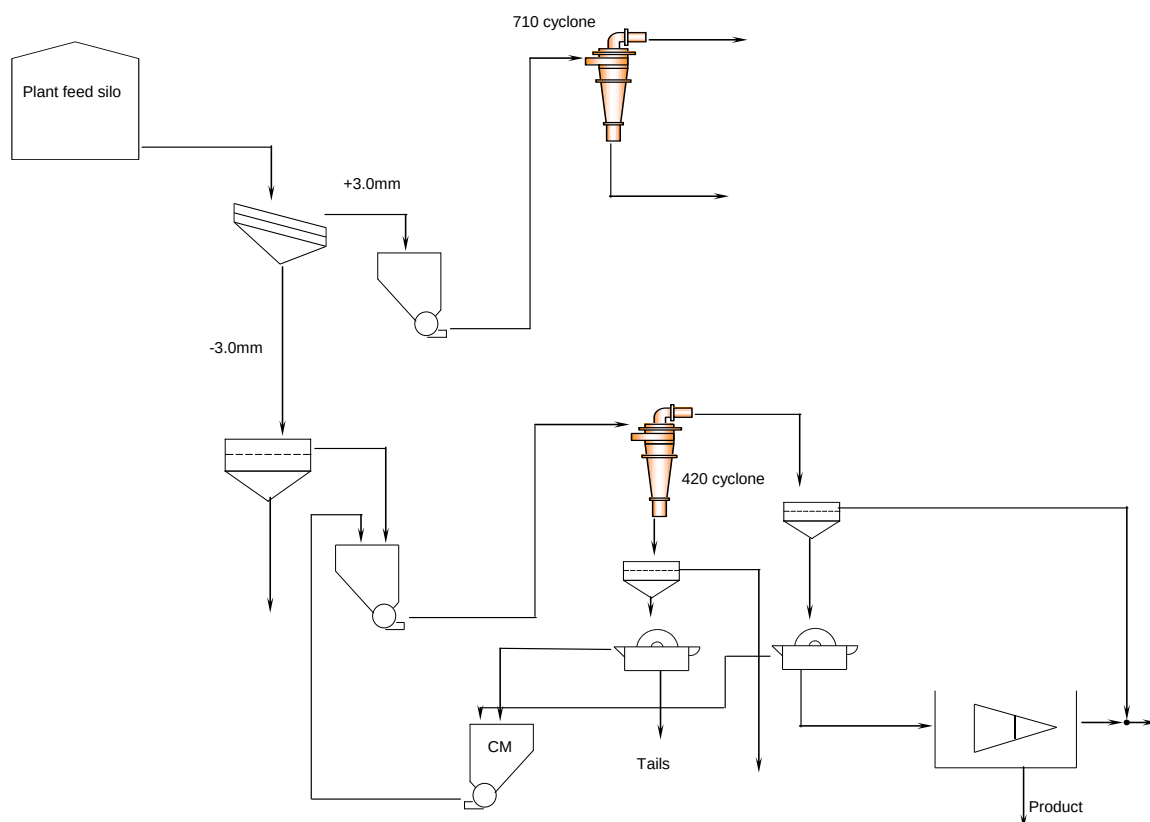


Figure 9.6: Scenario 5 process flow diagram

This circuit will closely resemble the circuit at Exxaro's Leeuwpan coal mine, except for drain and rinse screens before the magnetic separator, which are required to remove the oversize for the magnetic separators. The discard magnetic separator underflow will go to the flotation tailings system, and the product magnetic separator underflow will go to the centrifuge. The spiral feed dewatering screens can also be used if necessary. The -3mm fraction will be 72 t/hour and will consist of two 40 t/hour modules to correspond with the Tshikondeni plant feed rate. Extra capacity will be created for the 710mm cyclones because more material is diverted to the fines circuit.

9.1.7 Experimental Test Procedures

Samples of plant from Mutale Shaft and Vhukati Shaft were taken in March 2007 and subjected to a detailed particle size distribution by screening at 13.2mm, 9.5mm, 6.7mm, 4.75mm, 3.35mm, 2.36mm, 1.7mm, 1.4mm, 0.85mm, 0.6mm, 0.3mm, 0.212mm, 0.15mm and 0.075mm. Thereafter, classes were combined to give the following samples for float and sinks analysis: +6.7mm, -6.7+3.35mm, -3.35+1.4 mm, -1.4+0.6mm, -0.6+0.212mm, -0.212+0.075mm. Each of these samples were then subjected to float and sinks testing over a relative density range of 1.30-1.80 in 0.05 density intervals. The ash content of each density fraction, including the sinks 1.80, was determined. The detailed results are presented in Appendix 8.

9.2 Results and Discussion

9.2.1 Screening Results

Figure 9.7 shows that similar particle size distributions were obtained from both shafts, though Vhukati contained a slightly higher proportion of -1.4mm material at approximately 34%, compared to Mutale at approximately 28%. These values are within the normal ranges experienced from historical plant data, and close to the plant design specification of 30% passing 1.4mm.

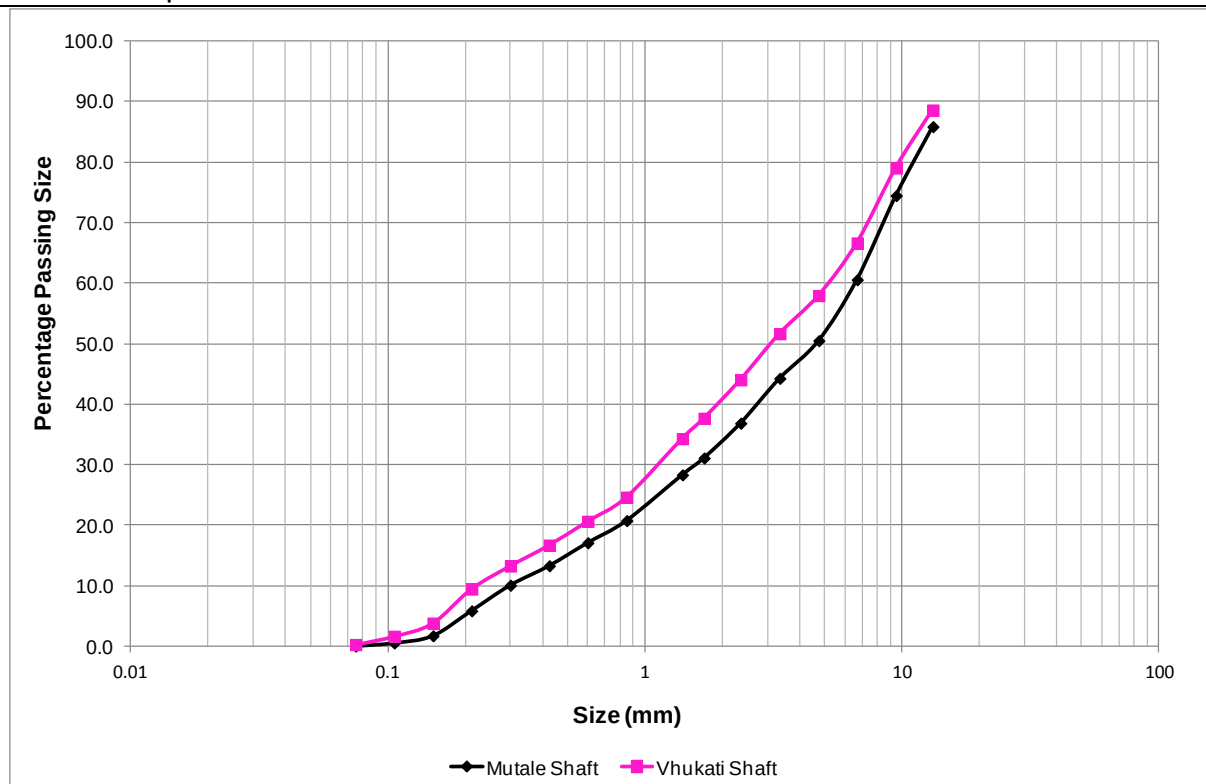


Figure 9.7: Particle size distribution for Mutale Shaft and Vhukati Shaft

9.2.2 Washability Results

9.2.2.1 Mutale Shaft

Figure 9.8 and Figure 9.9 clearly show that the -0.212mm fraction is the best coal resulting in the highest yield at 14% ash. It is also illustrated that the different relative densities to which each size class must be washed to achieve the correct ash value in the product. It differs from a relative density of 1.75 for the $-1.4+0.6\text{mm}$ coal to a significantly lower relative density of 1.40 for the $-6.7+3.4\text{mm}$ fraction.

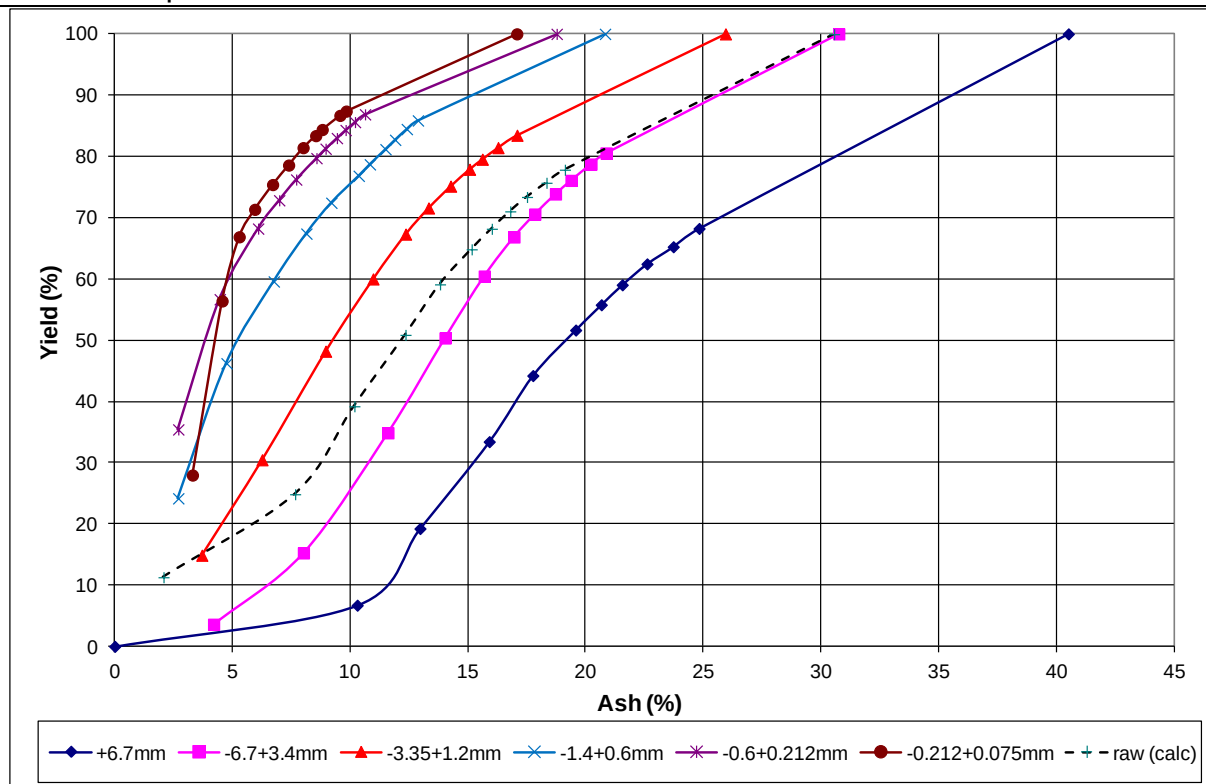


Figure 9.8: Yield vs. ash relationship for Mutale Shaft

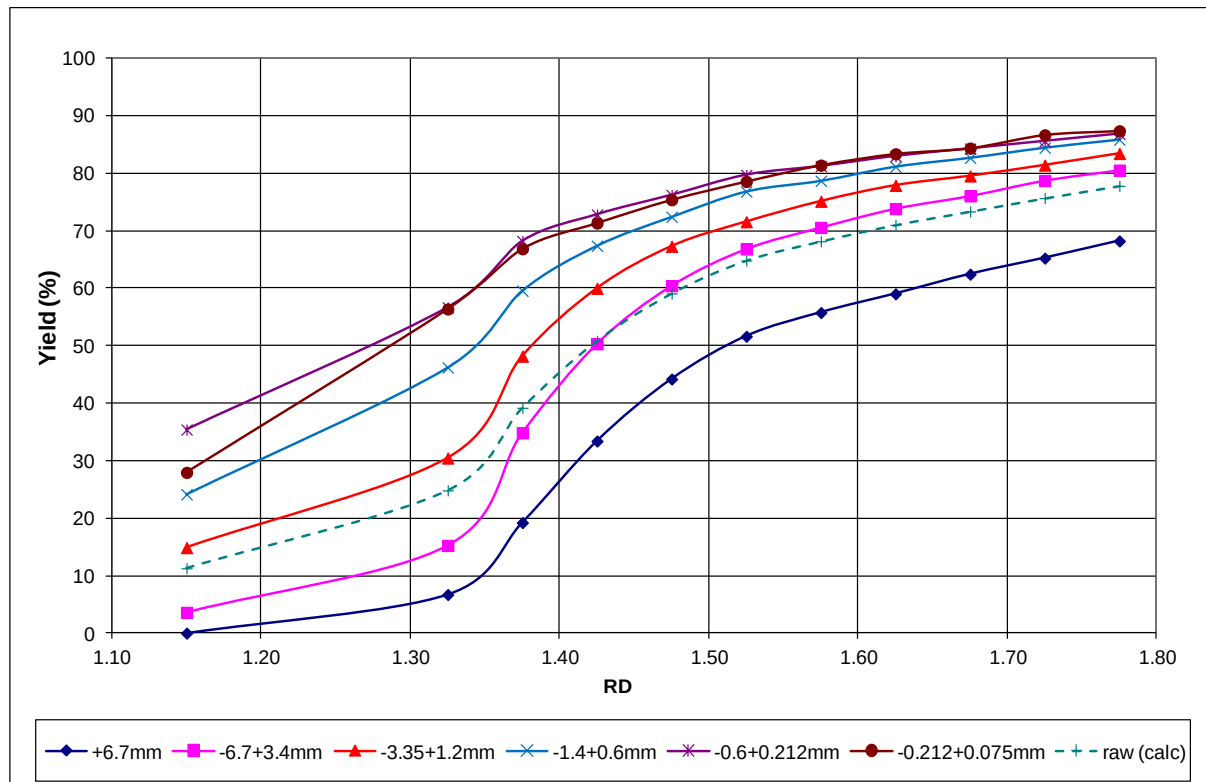


Figure 9.9: Yield vs. relative density relationship for Mutale Shaft

The same trend can be seen for the Vhukati washability data as for Mutale, with the exception that Vhukati has lower total ash values. Vhukati has raw coal ash content of approximately 28% ash compared to Mutale at 31% ash.

Figure 9.10 and Figure 9.11 clearly show that the -0.212mm fraction is the best coal resulting in the highest yield at 14% ash. It is also illustrated that the different relative densities to which each size class must be washed to achieve the correct ash value in the product. It differs from a relative density of 1.75 for the -1.4+0.6mm coal to a significantly lower relative density of 1.45 for the -6.7+3.4mm fraction.

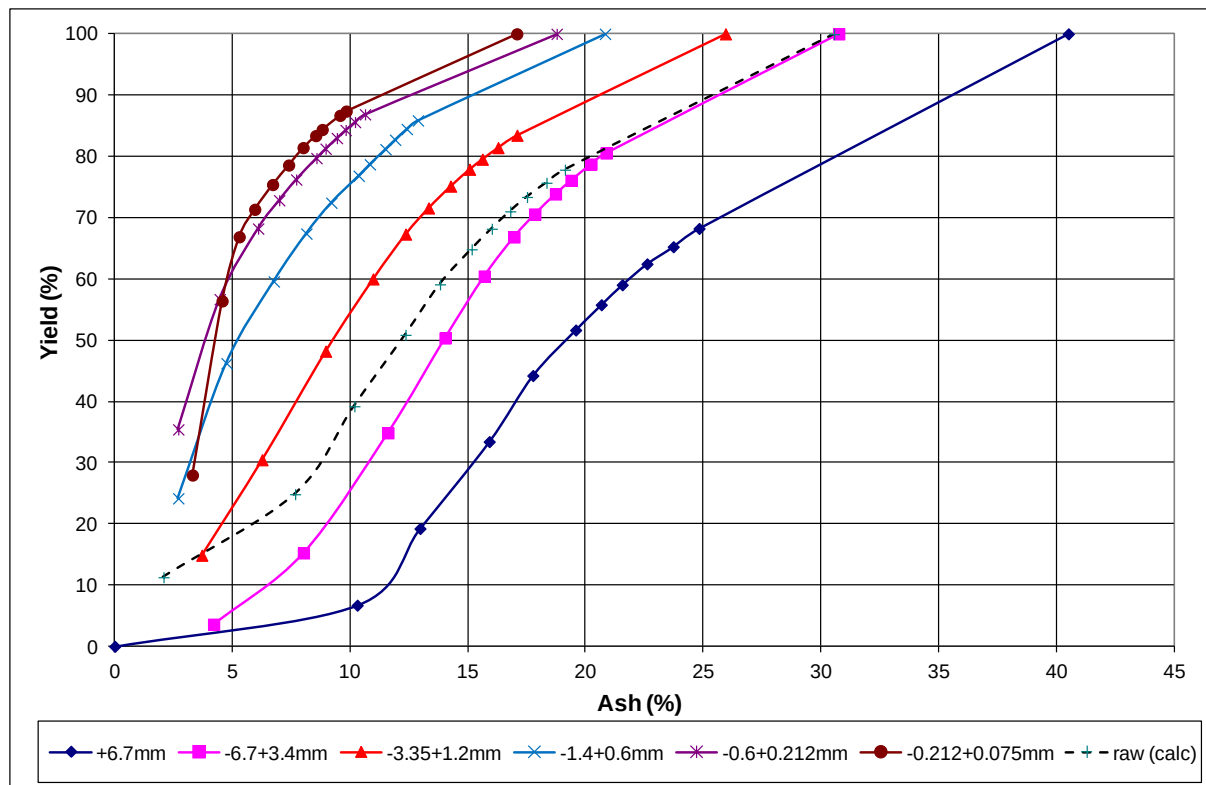


Figure 9.10: Yield vs. ash relationship for Vhukati Shaft

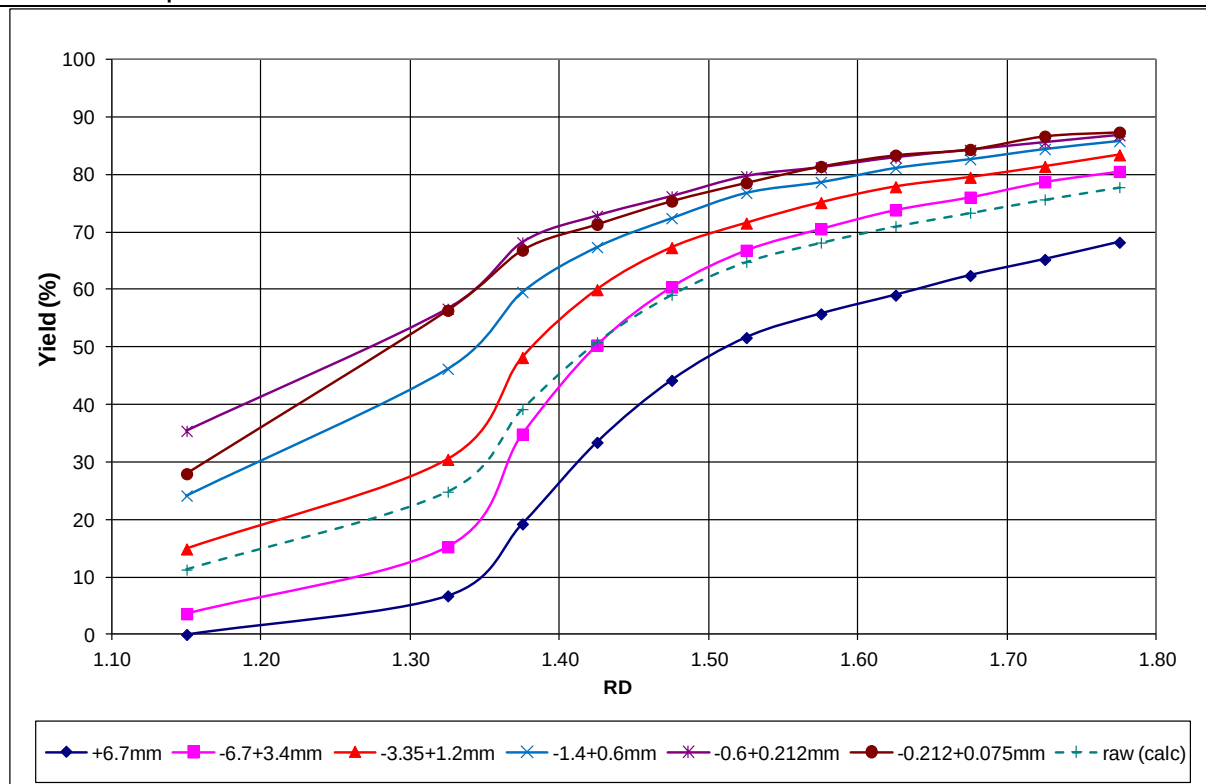


Figure 9.11: Yield vs. relative density relationship for Vhukati Shaft

9.2.2.3 Comparison of the Washability Curves

A comparison of the yield versus relative density relationships for both shafts is illustrated in Figure 9.12, which shows that Vhukati Shaft will produce higher yields for any given ash content compared to Mutale Shaft. For a 14% ash product, slightly different separation densities will be required with Mutale Shaft at 1.48 and Vhukati at 1.51. Although both coals display the same trend for the washability curves, this confirms the current plant operational philosophy of washing each shaft separately maximises the product yield.

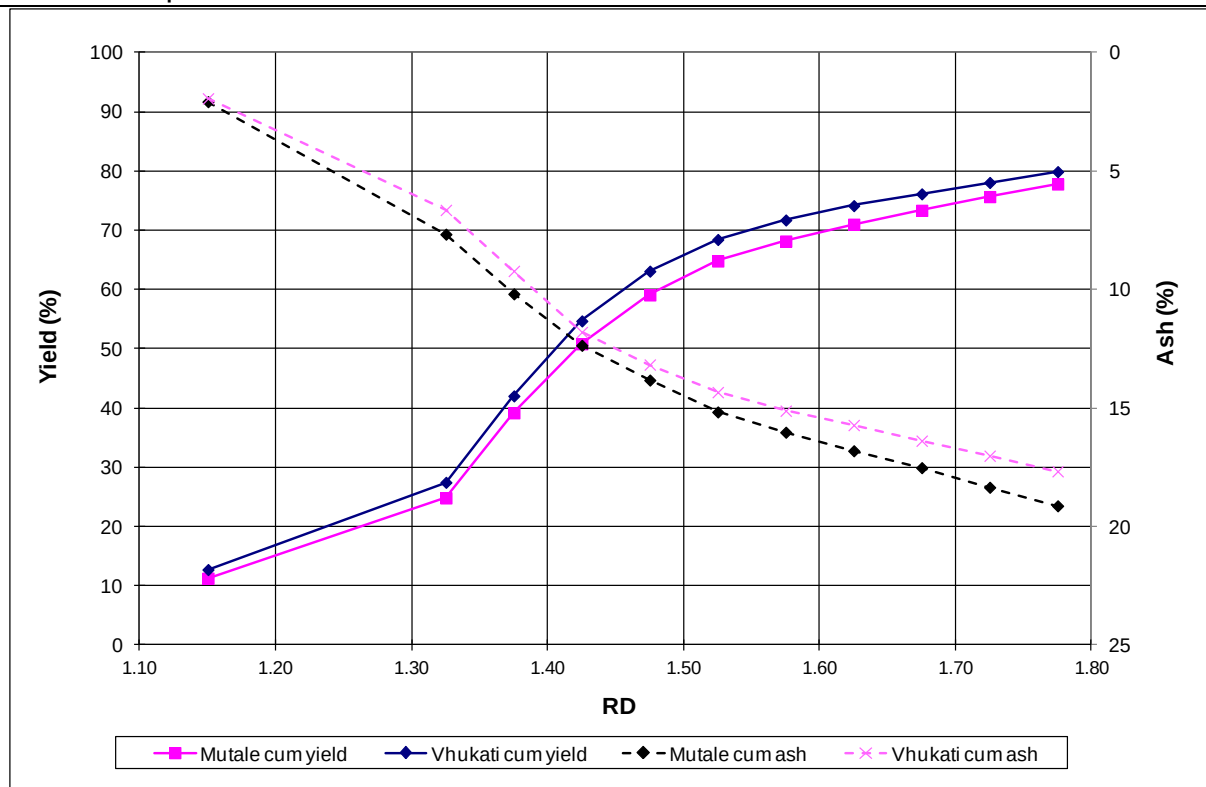


Figure 9.12: Yield vs. relative density comparison for Mutale and Vhukati Shafts

9.3 Simulation and Evaluation of the Scenarios

9.3.1 Simulation of the Scenarios

Van der Merwe (2007) performed LIMN simulations on the various scenarios which were also checked using the Coaltech model developed by de Korte (2007). The results are detailed in Appendix 7 and summarised in Table 9.1 and Table 9.2 for Mutale Shaft and Vhukati Shaft respectively.

With the exception of Scenario 5, all alternatives considered indicated that some gain in yield would be achieved, however the greatest improvement would be noted for Mutale Shaft which has previously been identified as difficult to process in the flotation section of the plant. For Mutale, gains of up to 3% yield may be possible by the reconfiguration of the plant to Scenario 4, i.e. processing the +1.4mm material in

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the existing 710mm diameter DMS cyclone, processing the -1.4+0.212mm material in a newly installed 420mm diameter DMS cyclone, and the -0.212mm material by the existing froth flotation plant.

Table 9.1: Summary of Mutale Shaft simulations (Van der Merwe, 2007)

Scenario	Description	-212mm			-1.4mm (-3mm)			+1.4mm			Total	
		Ash %	Yield %	Mass % of feed	Ash %	Yield %	Mass % of feed	Ash %	Yield %	Mass % of feed	Ash %	Yield %
0	Base case											
1	All to 710 cyclone										14.0	57.3
2	-1.4mm to new 420 cyclone +1.4mm to current 720 cyclone				11.6	75.6	28.3	15.5	50.3	71.7	14.0	57.5
3	-0.212mm to flotation +0.212 to 710 cyclones	10.0	85.0	5.9				14.3	57.0	94.1	14.0	58.7
4	-0.212mm to flotation 1.4+0.212 to 420 cyclones +1.4mm to 710 cyclone	10.0	85.0	5.9	12.0	77.1	22.4	15.5	52.8	71.7	14.0	60.1
5	-3.0mm to 420 cyclones +3.0mm to 710 cyclone	0.0	0.0	0.0	11.9	68.4	44.3	16.6	47.5	55.7	14.1	56.8

Table 9.2: Summary of Vhukati Shaft simulations (Van der Merwe, 2007)

Scenario	Description	-212mm			-1.4mm (-3mm)			+1.4mm			Total	
		Ash %	Yield %	Mass % of feed	Ash %	Yield %	Mass % of feed	Ash %	Yield %	Mass % of feed	Ash %	Yield %
0	Base case											
1	All to 710 cyclone										14.0	66.5
2	-1.4mm to new 420 cyclone +1.4mm to current 720 cyclone				11.0	81.9	34.3	16.3	59.0	65.7	14.0	66.9
3	-0.212mm to flotation +0.212 to 710 cyclones	10.0	90.0	9.5				14.4	64.3	90.5	14.0	66.7
4	-0.212mm to flotation 1.4+0.212 to 420 cyclones +1.4mm to 710 cyclone	10.0	90.0	9.5	12.0	84.5	24.8	16.0	56.8	65.7	14.0	66.8
5	-3.0mm to 420 cyclones +3.0mm to 710 cyclone	0.0	0.0	0.0	11.4	76.4	51.7	18.1	55.6	48.3	14.1	66.4

Table 9.3 illustrates the expected yields and feed compositions that were used in the financial model (Van der Merwe, 2007). The total yield for each shaft and expected run of mine were used to calculate a weighted average plant yield for each year of the life of mine plan. This plant yield was used to calculate expected product tons per year for the duration of the life of mine planning. The life of mine plan used for these calculations is the approved plan as received from the Technical Services department of Tshikondeni Coal during April 2007. An overall increase of between 3% and 4% was noted for all scenarios relative to the base case as can be seen in Table 9.4.

Table 9.3: Average plant yield for the life of mine per scenario simulations (Van der Merwe, 2007)

Description	Shaft	Ultrafine		Fine		Coarse		Total Yield (%)
		Mass (%)	Yield (%)	Mass (%)	Yield (%)	Mass (%)	Yield (%)	
Base case	Vhukati	-	-	34.4	82.0	65.6	54.7	64.1
	Mutale	-	-	28.3	65.0	71.7	47.9	52.7
Scenario 1	Vhukati	-	-	-	-	100.0	66.5	66.5
	Mutale	-	-	-	-	100.0	57.3	57.3
Scenario 2	Vhukati	-	-	34.3	86.5	65.7	54.7	66.9
	Mutale	-	-	28.3	77.4	71.7	47.9	57.5
Scenario 3	Vhukati	9.5	90	-	-	90.5	64.3	66.7
	Mutale	5.9	85	-	-	94.1	57.0	57.5
Scenario 4	Vhukati	9.5	90	24.8	84.5	65.7	56.8	66.8
	Mutale	5.9	85	22.4	77.1	71.7	52.8	60.1
Scenario 5	Vhukati	-	-	51.7	76.4	48.3	55.6	66.4
	Mutale	-	-	44.3	68.4	55.7	47.5	56.8

Table 9.4: Average plant yield for the life of mine per scenario simulations (Van der Merwe, 2007)

Description	Average plant yield for life of mine
Base case	59.2%
Scenario 1	62.6%
Scenario 2	62.8%
Scenario 3	63.3%
Scenario 4	64.0%
Scenario 5	62.3%

9.3.2 Financial Evaluation of the Scenarios

Van der Merwe (2007) indicated that the industry norm for dense medium separation is R5/tonne feed and for flotation R20/tonne feed, which represented a ratio of 1:4. Tshikondeni's operating expense for the beneficiation plant is R17/tonne feed based on plant actual figure during 2007. These values were adjusted to represent Tshikondeni's actual values in the base case scenario and are presented in Table 9.5. The ratio between flotation and dense medium was kept constant at 1:4. The operating value for the fine dense medium beneficiation is slightly increased to make provision for expected higher magnetite consumption. Values of R3.91 and R3.65

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were used during 2006 for the financial evaluation of dense medium fine coal beneficiation at two of Exxaro's other operating business units.

Table 9.5: Operating and Capital expense figures used in the financial evaluation simulations (Van der Merwe, 2007)

Description	Shaft	Ultrafine		Fine		Coarse		Feed R/ton	Average R/ton	Capex Rand
		Mass (%)	R/ton	Mass (%)	R/ton	Mass (%)	R/ton			
Base case	Vhukati	-	-	34.4	34.00	65.6	8.50	17.27	16.49	0
	Mutale	-	-	28.3	34.00	71.7	8.50	15.72		
Scenario 1	Vhukati	-	-	-	-	100	8.50	8.50	8.50	4809350
	Mutale	-	-	-	-	100	8.50	8.50		
Scenario 2	Vhukati	-	-	34.3	10.00	65.7	8.50	9.01	8.97	6847750
	Mutale	-	-	28.3	10.00	71.7	8.50	8.92		
Scenario 3	Vhukati	9.5	34.00	-	-	90.5	8.50	10.92	10.46	4490850
	Mutale	5.9	34.00	-	-	94.1	8.50	10.00		
Scenario 4	Vhukati	9.5	34.00	24.8	10.00	65.7	8.50	11.29	10.82	7229950
	Mutale	5.9	34.00	22.4	10.00	71.7	8.50	10.34		
Scenario 5	Vhukati	-	-	51.7	10.00	48.3	8.50	9.28	9.22	8440250
	Mutale	-	-	44.3	10.00	55.7	8.50	9.16		

Scenario 1 has the lowest capital expenditure, R4.8 million, as well as the lowest operational cost with R8.5/tonne feed. Scenario 5 will be the most expensive with a capital estimation of R8.4 million. The capital used was as per a similar unit within Exxaro (R10.8 million).

The financial model results as determined by Van der Merwe (2007) are presented in Table 9.6:

Table 9.6: Financial evaluation results simulations (Van der Merwe, 2007)

Description	Average Sales Price (Rand)	Difference From Base Case (%)	Tonnes Produced Over Life Of Mine (MT)	NPV (R Millions)	IRR (%)
Base Case	32.19	0	3.44	0	0
Scenario 1	16.97	47.3	3.63	11.4	67
Scenario 2	18.3	43.2	3.65	8.5	46
Scenario 3	20.28	37	3.67	6.8	52
Scenario 4	21.41	33.5	3.71	3.5	29
Scenario 5	19.36	39.9	3.61	6.4	36

The average sales price per product tonne was calculated as follows:

$$\text{Average Sales Price (R/T)} = \frac{(\text{Operating Cost} + \text{Depreciation} + \text{Royalties} + \text{Exxaro Management Fee})}{\text{Product Tonnes}}$$

where the "Exxaro Management Fee" is a percentage of the operating costs plus depreciation and royalties as agreed with ArcelorMittal South Africa (and cannot be published here due to confidentiality reasons). The reason for this management fee

is that Tshikondeni is an ArcelorMittal South Africa asset but operated and managed by Exxaro on their behalf.

From Table 9.6 above it is evident that Scenario 1 (wash to zero in the existing 710mm diameter cyclones) would have given the largest saving at 47% of the base case as well as the most favourable NPV and IRR values. Scenario 5 had the highest capital expenditure requirement whilst Scenario 4 was the highest in terms of operational expenses. In terms of yield realization, Scenario 5 would be the lowest, whilst Scenario 4 would return the highest product yield.

Scenario 2 was, however, considered the best option to further pursue due to several reasons:

- It is a proven technology and a similar plant was being constructed at Exxaro's Leeuwpan coal mine.
- The influence of the lower operational expenses was significant, and only Scenario 1 had a bigger influence.
- The separate -1.4mm circuit will provide control over the final product quality.
- An increase in yield from the base case will be achieved
- It will be easy to implement within the current Tshikondeni plant and operations.

9.3.3 Conceptual Design Scenario 2

Based on the above evaluations, a conceptual design was drawn up by the draughtsman located at Exxaro's Metallurgy Research and Development facility, where it was considered that the circuit for the 420mm dense medium cyclone could be easily retro-fitted into the area occupied by the decommissioned spirals plant. However, should the project have continued to a more detailed design phase, a standalone circuit would also have been considered for completeness.

Selected views of the conceptual design are presented in Figure 9.13 to Figure 9.15.

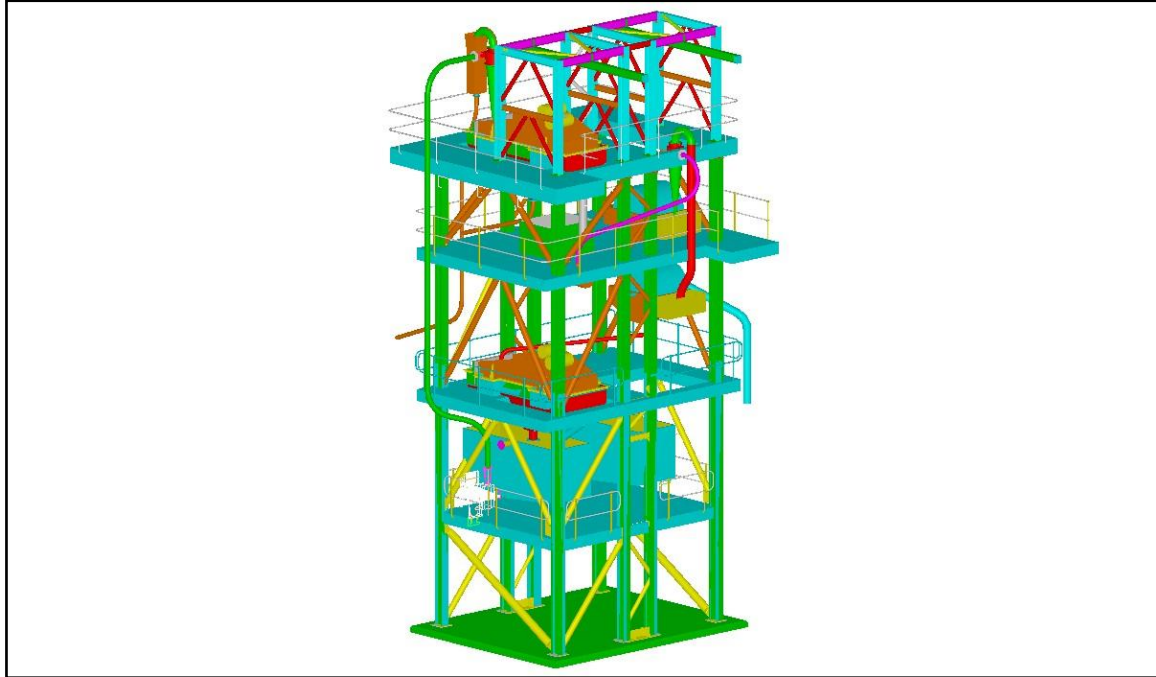


Figure 9.13: Conceptual design showing placement of equipment in the spiral structure alternative view 1

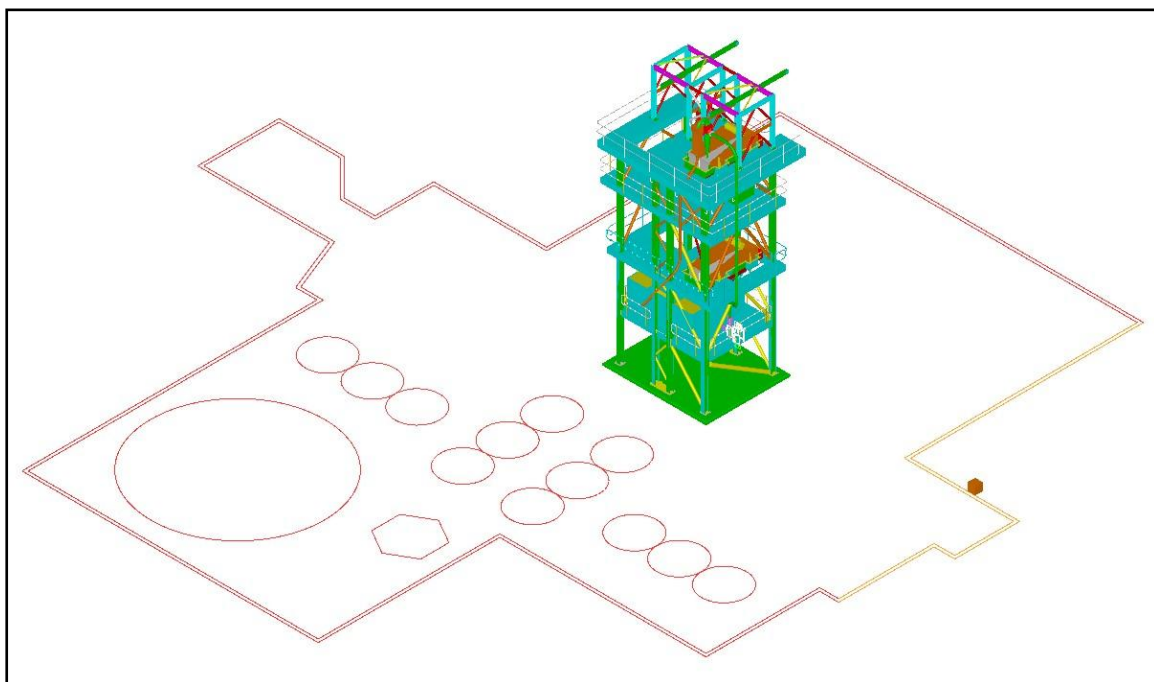


Figure 9.14: Conceptual design showing placement of equipment in the spiral structure alternative view 2

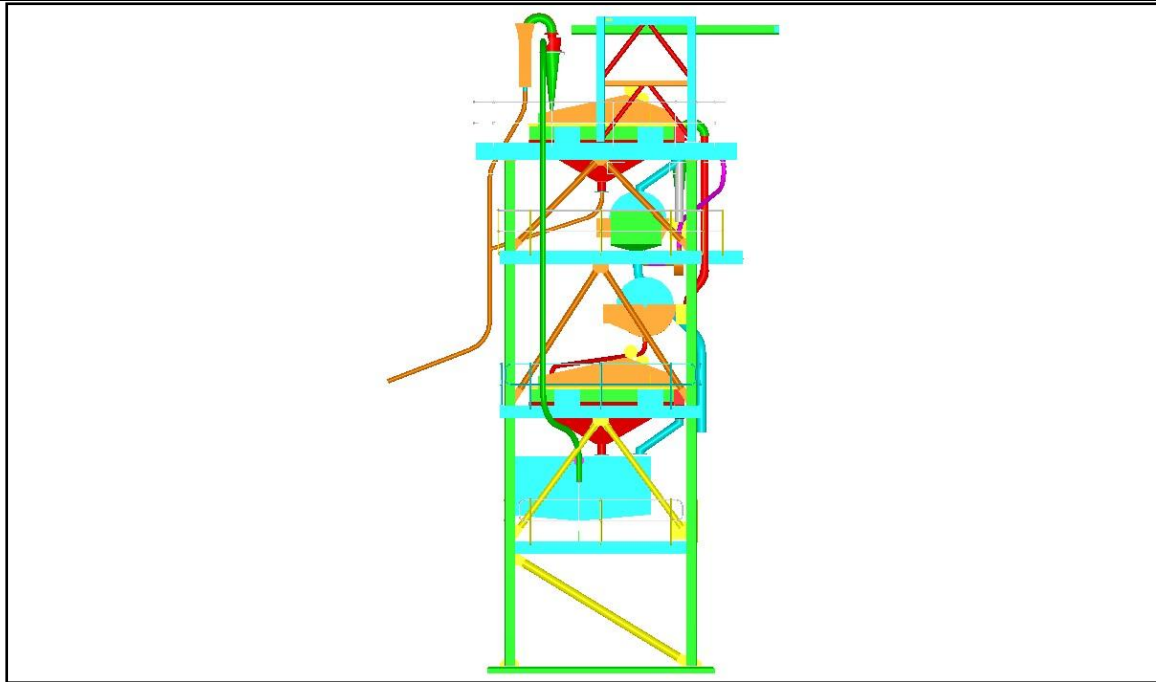


Figure 9.15: Conceptual design showing placement of equipment in the spiral structure

The project was scheduled to go to a detailed feasibility level study during 2008 to improve the capital and operating costs. Additional test work was also to be undertaken to consider more samples to better represent the various sources of coal being fed into the plant to further refine the flowsheet. However, the project was stopped due to the economic downturn which began in 2008 and severely affected the steel industry, and particularly ArcelorMittal South Africa who would essentially provide the capital required for a project of this nature. Moreover, given the short life of mine, with closure estimated for early 2015, if the coal price did recover sufficiently enough, the yield gains would not justify the increased project costs due to inflation.

9.4 Reconfiguration of the plant by replacement of the existing dense medium cyclones with a three product cyclone

The concept of using a three product cyclone within the South African coal processing fraternity in the Witbank Coalfields is gaining acceptance as discussed in Chapter 4. The application at Tshikondeni is most likely not feasible given that the potential yield for a power station grade coal is low, and the current sales price in the order of R120 (Martin, 2014) would not justify the reconfiguration of the plant, nor would a return on the investment be realised. It is also noted that the transport costs from Tshikondeni to one of the power stations would be in the order of between R500 and R600 per tonne (Sibanyoni, 2014).

From Chapter 4, it was also found that a power station yield of 17-23% at acceptable ash levels was achieved on material with a topsize of 8mm, which is unlikely to separate well in a three product cyclone once the breakaway size is considered, likewise the full discard stream at a nominal 13mm topsize.

9.5 Chapter Summary

A variety of alternative flow sheets were evaluated with a view to improving the plant yields.

The reconfiguration of the plant to allow for the installation of a fine coal dense medium cyclone treating material in the range of $-1.4+0.212\text{mm}$ would have enabled losses to the discard streams to be prevented, and resulted in an increase in yield of approximately 4%. However, global economic factors prevented this study progressing to a detailed design phase.

A conceptual design showing the installation in the decommissioned spirals plant was presented

The three product cyclone would not be suitable for installation at Tshikondeni.

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Chapter 10: Summary of the Findings

This Chapter presents a consolidation of all the findings from the various sections of the work as discussed in the preceding Chapters.

10.1 Chapter 2 Findings

The initial petrographic examination of the plant discard revealed that it contained approximately 47% vitrinite and a reflectance value of 1.32, which if economically recoverable would be suitable for supply as a hard coking coal product. At a relative density of 1.80, the vitrinite content was found to be above 60%. Examination of the fractional results, showed that vitrinite contents above 40% were noted up to a relative density of 1.60.

Plant efficiency tests by size revealed that the dense medium cyclone lost efficiency below 2.8mm and that any losses to the discard would tend to be to the finer fractions. The discard was also noted to contain a significant proportion of larger, slab like material. The cyclone feed contained 3% of material finer than 1.4mm indicating good performance of the desliming screen.

The monthly discard composite samples were examined in terms of the washability of the +1.4mm material and it was noted that very little losses were being experienced at the lower density fractions. The flotation testing on the -1.4mm material showed losses of 7.2% in mass terms at an ash of 14%. However, subjecting the -1.4mm material to a float and sinks procedure revealed that approximately 40% of the mass could be recovered at an ash of 14% and relatively low separation relative density in the order of 1.52-1.53.

The flotation circuit efficiency was therefore further examined in terms of size, and several samples from each of the three shafts examined. It was found that the losses in this section were to the +0.6mm fraction.

From this chapter it is was evident that losses occurred at the finer sizes of the dense medium section and coarser sizes of the flotation section of the plant. Discard retreatment or reconfiguration of the plant circuit could therefore improve the overall recovery of hard coking coal at Tshikondeni.

10.2 Chapter 3 Findings

A total of 30 auger borehole samples were extracted from the two discard dumps and the Visgat Dam, of which 24 samples came from the old dump, 4 samples from the current operational dump, and 2 from the Visgat Dam respectively.

The individual boreholes were divided into 4.5m long sub-samples to represent a typical mining bench and each subsample examined for the in-situ ash content, which was found to approximately 50% in the old dump, 55% in the current dump and 24.5% in the Visgat Dam.

A three dimensional model showing the ash distribution in the old dump was developed along with the graphical representations of the ash distribution through the dump in consecutive 4.5m benches.

10.3 Chapter 4 Findings

In this section of the work, a literature survey with respect to destoning of coal was undertaken and from that technologies considered appropriate for the upgrading of the Tshikondeni discards were examined.

Five one tonne bulk samples were collected from the three operating shafts, viz. two from Mutale, two from Vhukati and one from Goni. The ash content of these samples was determined to be between approximately 56-61%.

From the washability by size fraction exercise it was noted some destoning could be achieved by screening out the +8mm, which contributed to between 53-58% of the mass in the samples at ash contents of between 65-70% ash. The -8mm fractions has ash contents of between approximately 46-50%.

Simulation of a dense medium separation on the -8+0.5mm fraction showed that a coking coal specification of 14% ash could be met at separation relative densities IN the order of 1.49-1.53 though the mass yields were very low at between 6-14%. A vitrinite content in the order of 79-83% would also be expected from this product.

Given that the average plant yield is typically in the order of 55%, the yield from rewashing this fraction would equate to approximately 3-8% of the run of mine, and is unlikely to produce a return on the processing costs.

A further simulation was performed at a separation relative density of 1.80g/cm^3 , where it was found that product ash contents of between approximately 24-29% could be achieved. The vitrinite content would be expected to be approximately 60%. The mass yields were also noted to be low at between 22-26%, which equated to approximately 8-10% on a run of mine basis. Despite the higher volume of material available for sale, the revenue for this product would be substantially less than that for hard coking coal, and the transport costs alone would outweigh the costs of production since it would have to be trucked approximately 500km to the nearest power station.

It was found that the an air jig such as the FGX Separator would not be suitable for the discard at Tshikondeni, however, dense medium separation would be very applicable. Should this have worked, it would have been an ideal process due to the severe water scarcity in the Tshikondeni area, which has approximately 300mm rainfall per annum based on the historical records from the Tshikondeni Survey department.

The use of the Reflux classifier to treat the fines in the range $-1.4+0.6\text{mm}$ was not considered due to the short life of mine by the time the unit had been fully commissioned at Exxaro Research and Development (late 2012).

Overall, it was noted that destoning could be effected by the application of dense medium, with or without the use of screening. From this section of the work, the destoned product could be milled to create liberation of the coal from the waste rock, and thereafter froth flotation employed to produce the hard coking coal product desired by ArcelorMittal South Africa.

10.4 Chapter 5 Findings

The five sub-samples from the destoned discards from the three operating shafts were subjected to the Mintek laboratory scale batch ball mill test procedure which included a process simulation.

From the testwork conducted it was found that the five samples tested were amenable to milling to 80% passing 212 μ m and would be achieved at specific energy inputs of between 6.22-6.99kWh/t, whilst the circulating loads would be expected to vary between 118-133%.

The energy input is low relative to many minerals as indicated in the standard minerals processing text books (Kelly and Spottiswood, 1982; Wills and Napier-Munn, 2006), and the Tshikondeni discard after destoning would therefore be expected to produce a feed for froth flotation.

10.5 Chapter 6 Findings

The initial phase of the test programme was undertaken on the auger borehole core samples after screening destoning and milling to -212 μ m. These samples were subjected to the standard Tshikondeni flotation rate test using the standard reagent suite comprising 85% collector (paraffin) and 15% frother (Betafroth BTN3), which was dosed at a relatively low dosage equating to approximately 0.7kg/t in the flotation circuit.

An average mass pull (yield) of 25.5% at an average ash content of 19.3% was observed for the old dump, whilst for the current dump the average mass pull and ash content were noted to be 18.8% and 20.3% respectively. The standard test showed that little selectivity was experienced in the flotation test, and the first concentrate (0-30 seconds) contained most of the mass and ash contents above 15%. The high ash content could potentially be improved by reagent optimization and/or the addition of a cleaner stage. Several samples along with the plant feed and plant discard should have been subjected to a release analysis to gain a better understanding of the flotation process and eliminate the effects of entrainment.

The flotation rate information from application of the Klimpel and Kelsall flotation models indicated the dumps to be slow floating.

The samples from the Visgat Dam also produced high ash contents overall, though it was noticed that the lower ash material floated later in test, contrary to what would be normally expected. Again, tailoring of the process in terms of reagents and cleaner flotation may positively influence the behaviour of this material. The modelled flotation rate constants also indicated slow flotation characteristics

Tests were performed on the arising discard composites from four separate months and again the specification ash content was not reached and varied between 20-23% at yields of between 18.5-23.1%. Milling of the material appeared to marginally improve the overall yield obtained, but further work to optimize the process, particularly in terms of reagent selection would be required.

The 20 tonne bulk sample taken from the old discard dam was subjected to more extensive testing, and included the combined sizing followed by dense medium separation considered in previous chapters.

For the -3mm material, which was destoned at a slightly lower relative density of 1.75, yields of 69% to 74% were achieved in a rougher stage of flotation at ash contents of approximately 19%, using slightly different reagent suites. These rougher concentrates were then combined and subjected to a cleaner stage, which gave a yield of approximately 84% at an ash content of 16.8%.

It was noted that the use of organic liquids in the destoning process appeared to influence the flotation performance: lower product qualities and minimal response in terms of selectivity were observed. A short study comparing the use of organic liquids to a suspension of magnetite and water was performed, where it was found that the use of organic liquids positively affected the yield by between 12-15% and increased the product ash by up to 2%.

Further work was therefore undertaken on the discard at various sizes and destoning relative densities and alternative reagent suites, where it was noted that the removal of the finer sizes (-6mm or -3mm) would give improved concentrate ashes though

lower yields. Better results could possibly be achieved by further testwork to optimise the destoning and flotation test programmes.

On the -0.5mm fraction only, it was observed that concentrate ashes of close to the 14% specification could be achieved from cleaning the rougher concentrate, but at fairly low overall yields.

10.6 Chapter 7 Findings

After the destoning, milling and flotation of the discard, the flotation product would need to be dewatered, and due to the fine size, most likely require briquetting to enable ease of handling.

The existing solid bowl centrifuges are capable of dewatering the froth flotation product effectively, and the existing plant potentially has surplus capacity which could potentially be used to dewater the froth flotation product from a discard retreatment plant.

Tshikondeni coal is very amenable to binderless briquetting, and the application of this technique may be required if a discard retreatment plant is installed to enable the additional froth flotation product to be easily handled.

10.7 Chapter 8 Findings

A detailed financial evaluation of the project was undertaken using the Project Evaluation Tool developed by Exxaro's Corporate Finance Department. The original work from 2009 was presented along with an update for 2015.

The capital requirement for the plant was kept at a relatively low value since the project could be undertaken by modification of the plant after the ROM sources had been depleted.

In both versions, the project exceeded the hurdle rates as per requirements, and the payback period in real terms would be just under 2 years for both the original 2009

and updated 2015 evaluations. This makes the project would make the project very viable for Exxaro. Unfortunately the project was not pursued further since it was not in line with the strategic plans for either Exxaro or ArcelorMittal SA.

Simplification of the plant also shown that the plant at 10% yield from a screening and dense medium process was also viable and paid back in marginally over 1 year. The flotation circuit would need some optimization to process the fines, and could add additional tonnes to the final product.

The unknown factors such as future Rand:US Dollar exchange rate, the hard coking coal price, and fuel and electricity costs though should not impact upon the project too severely the project.

10.8 Chapter 9 Findings

A variety of alternative flow sheets were evaluated with a view to improving the plant yields.

The reconfiguration of the plant to allow for the installation of a fine coal dense medium cyclone treating material in the range of $-1.4+0.212\text{mm}$ would have enabled losses to the discard streams to be prevented, and resulted in an increase in yield of approximately 4%.

Economic factors prevented this study progressing to a detailed design phase.

A conceptual design showing the installation in the decommissioned spirals plant was presented

The three product cyclone would not be suitable for installation at Tshikondeni.

Chapter 11: Conclusions and Recommendations

This Chapter presents the overall conclusions and recommendations of the project based on the findings from the various sections of the work.

11.1 Conclusions

Tshikondeni Coal was the only hard coking coal operational in South Africa at the time of the study, and therefore of strategic importance to both Exxaro and ArcelorMittal South Africa, who use it as the only local hard coking coal input into their process (the other 3 are sourced from Australia and New Zealand), and that any extension of this supply would be beneficial for both parties.

From the work performed, the major conclusions that could be drawn from this project in terms of the original five objectives set in the Introduction chapter in Section 1.1 are presented below.

11.1.1 Objective 1

Objective 1 was stated as: *To determine the operational efficiency of the plant and identify any possible losses of saleable product to the discard streams.*

In terms of this objective, it was found that:

- Through a process of auditing the various processes in the plant, sources of loss of hard coking coal were identified, and it was found that the discard contained significant amounts of the maceral vitrinite.
- Losses from the dense medium cyclone were noted to be in the finer fractions, particularly in the -3mm fractions, where the cyclone efficiency was noted to “break away”.
- Losses from the flotation circuit were found to be in the +0.6mm size fraction.

11.1.2 Objective 2

Objective 2 was stated as: *Characterization of the discards dumps and the naturally arising discards with the aim to recover any potentially saleable product lost by the plant to the dumps.*

In terms of this objective, it was found that from a detailed auger borehole drilling exercise, five 1 tonne bulk samples from the naturally arising discards and a 20 tonne bulk sample from one of the dumps that:

- Saleable material was present within the two discards dumps and also the disused return water dam (the Visgat dam).
- A model geological model of the dump was able to be constructed showing the raw ash values in 4.5m layers, which would represent the benches used if the dump was to be mined.
- The testwork, and application of suitable technologies to recover the lost values in the discards was performed under the auspices of Objective 3

11.1.3 Objective 3

Objective 3 was stated as: *Perform an evaluation of various coal processing technologies for optimal flowsheet development for the design of a discards retreatment plant for the recovery of these losses.*

In terms of this objective, which looked in detail at the response of the auger borehole samples, 1 tonne bulk samples of naturally arising discards and a 20 tonne bulk sample in terms of recovery of saleable product, it was found that:

- A stepwise methodology based on various minerals processing unit operations was applied to the discards to see if the valuable hard coking coal could be separated from the true waste material:
 - It was found that by screening out of the +8mm material that it removed approximately 50% of the material at an ash content of approximately 65-70%. The -8mm could be further processed using dense medium, and that at the specification level of 14% ash low yields of between 6% to 14% could be achieved at relative densities in the order of 1.48-1.52. If the material was destoned at a relative density of 1.80, then the yield increased to approximately 20-22%, and would be suitable for the Eskom market, though the transport costs to the power utilities would not make this economical.
 - If the dense medium product after destoning at a relative density of 1.80 was subjected to milling and flotation, it may be possible to improve the quality. The -0.5mm material would be added into the mill without pre-concentration

- Milling specific energy inputs of between 6-7kWh/t would be required.
- Froth flotation of the material was difficult to get close to the required 14% ash content and also achieve acceptable yield, and would require extensive optimization should this route be followed.

11.1.4 Objective 4

Objective 4 was stated as: *Perform an economic evaluation of the proposed flowsheet for the discards retreatment plant.*

In terms of this objective, it was found that:

- A plant design, using the methodology applied under Objective 3, was developed and an economical evaluation was undertaken. The evaluation clearly showed a viable project with a positive NPV being achieved, both the IRR and MIRR exceeding the hurdle rate. Moreover, the project would require just under 2 years out of the 10 year project life span to payback, well below the minimum 3 years recommended by Exxaro's Corporate Finance Department.
- Simplification of the plant to allow for removal of +8mm material by screening and processing of the -8+0.5mm material in a dense medium cyclone also returned a favourable financial evaluation, which was calculated to be marginally over 1 year.
- Uncertainty of the Rand:US Dollar exchange rates, the hard coking coal price, and factors such as fuel and electricity, are a risk for the project.

11.1.5 Objective 5

Objective 5 was stated as: *To consider whether the existing plant could have been reconfigured to prevent or minimise the losses to the discards streams.*

In terms of this objective it was found that:

- The reconfiguration of the plant to accommodate the fines dense medium circuit would have reduced the losses experienced in the plant, and should

have been taken to detailed design level, despite the unfavourable economic conditions at the time.

- The economic evaluation of the project was noted to be very good and would have probably justified taking the project to the detailed design level, despite the unfavourable economic climate at the time of the study.

11.2 Recommendations

Based on the conclusions above, only two recommendations for the project exist:

- The potential for re-evaluating the project in the future still exists, and therefore the dump should be rehabilitated according to the mine closure plan, which is to cover it in soil for the plantation of grass. Should the dump be reconsidered in the future, the discards could still be recovered albeit at the expense of the removal of the top surface of grass and soil.
- Exxaro and ArcelorMittal South Africa should consider selling the dump to a third party to recover the values. This could be a potentially local economic development project for the community, which is sparse in terms of employment and lacking in terms of industrial development. The third party could potentially:
 - Reclaim the coking coal for supply to ArcelorMittal South Africa.
 - Supply the coal to the local brickmaking industry, the two current local brick makers are also relatively large employers of people in the area and the closure of Tshikondeni would have a negative effect upon these businesses both economically and socially, since there is no other local supply of coal discards which are used in their process.
 - Supply a lower grade of coal suitable for domestic use in stoves etc.
 - The use of the discards for independent power production for the locality, i.e. not via the Eskom route, though this would not be sustainable given the dump volumes, and would require the use of appropriate technology to set up this.
 - Any further work should be considered in terms of combustible recovery as well as both product yield and ash contents
 - Gasification is not a viable option as noted earlier in the introduction chapter.

Chapter 12: Contribution to the Discipline

Chapter 12 is the concluding Chapter of the thesis and is given to demonstrate what the candidate learnt during the project and the contribution that the Student made within the field of Chemical Engineering in terms of the Minerals Processing genre of the discipline.

12.1 Contribution to the Discipline

The candidate considered that the following contribution to the discipline of Chemical Engineering in terms of Minerals Processing was made:

1. Exxaro's Tshikondeni Coal was the only operating hard coking coal mine within the Republic of South Africa at the time of the study, and will have been either closed or in process of closure at the time of the submission of the thesis. Being the only source of this unique resource, it was therefore necessary to consider if the plant was operating efficiently, and whether or not any remaining values were present within the plant discards and if so, could they be extracted. The mine is also located in a very remote part of the country and is probably the major employer of the people residing close to the mine.
2. The initial work undertaken was essentially a plant audit, looking at several aspects of the plant operation, where it was found through a systematic evaluation of plant data that values were contained in the plant discards, and that these values were predominant in a specific size ranges due to the design of the plant. Firstly, the dense medium cyclone lost material below 3mm due to the size of the cyclone (710mm diameter) not being able to efficiently process material due to the effect of the so called "break away" size, i.e. the size at which the cyclone is unable to efficiently separate coal from waste. Secondly, the flotation circuit treats a -1.4+0mm size range and losses were noted to occur in the coarser size fractions.
 - a. This approach can be applied to virtually every operating plant and the loss of values to the discards can be determined readily.
3. The old dump was subjected to a detailed drilling programme using an auger drill to produce "borehole" cores which were analysed in 4.5m core lengths. A 3D geological model of the in-situ ash grades was made up based upon the data produced from the drilling programme. The borehole cores were also subjected to further and more detailed characterization to determine the extent of the values that could be recovered from the dump. The new dump could not be explored in as much detail due to safety requirements

associated with working on a dump, i.e. that has extensive earth moving vehicle activity upon it. The very fine material in the old return water dam, the Visgat Dam, was found to be of reasonable quality and could be either reprocessed or supplied to the local brick making factories as is.

- a. This approach can be applied to virtually any discard dump and used to evaluate the discards on the dump. The operational parameters of a mine can change and historical data may show significant losses have been included by the plant.
 - b. The candidate is aware that recovery of values from discard and tailings dumps has occurred in the past, particularly in terms of the gold dumps around the greater Johannesburg area, he is unaware of such an extensive exercise being applied to a coal dump.
4. The recovery of the values followed a logical and sequential evaluation of the various unit processes within minerals processing to upgrade the discards using simpler and cheaper options (screening and dense medium separation) ahead of the more complex and expensive (milling, flotation and briquetting) process routes available.
 - a. The route of screening ahead of dense medium separation, ahead of milling prior to froth flotation, and the application of a filter press and briquetting to the final product i.e. the concentrate from the froth flotation stage.
 - b. This sequence can be readily applied to coal discard dumps.
 - c. Froth flotation of coal usually occurs on current arising material, and therefore the application of milling prior to froth flotation is relatively an unknown event in coal processing plants. Extensive work was performed on five separate sources of discards and the specific energy required to mill the material to -0.212mm determined. Milling of the coal product delivered after processing is common in the power generation industry though.
5. A plant was designed to process the discards from the Tshikondeni plant, using the approach described in the previous point, though the economics were very favourable the project was not pursued further due to lack of strategic fit for both Exxaro and ArcelorMittal SA.

6. From point 2 above, as detailed in Chapter 9 of the thesis, some parallel work was undertaken to rectify the losses by the reconfiguration of the plant which would ensure that the finer sizes of the existing dense medium cyclone and the coarser sizes of the froth flotation sections were processed in a more appropriate manner. The work was included for overall completeness given that Tshikondeni Coal is unique as the only hard coking coal producing mine in the country at the time, and hence needed to ensure that losses to the discards could be prevented. The prevailing economic conditions at the time did not allow for the project to go further, unfortunately.
 - a. By identifying the plant losses, any plant can be corrected by reconfiguration of required allowing it to perform more efficiently and allow for maximum resource extraction of a non-renewable source.

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Appendix 1: Quantifying the Losses

This appendix contains the data used to quantify the losses of plant product to the plant discards as discussed in Chapter 2 of the main body of the thesis.

Test data is presented for plant efficiency testing and the effect of particle size in the flotation circuit.

A1.1 Plant Efficiency Testing

The results from the test examining the effect of feed size on DMS cyclone efficiency performed on the 2nd August 2007 are summarized below.

Table A1.1: Test screening data for the cyclone feed, product and discard

Screen Size (mm)	Feed			Product			Discard		
	Mass (%)	Cumulative Retained (%)	Percentage Passing Size	Mass (%)	Cumulative Retained (%)	Percentage Passing Size	Mass (%)	Cumulative Retained (%)	Percentage Passing Size
16	22.1	22.1	77.9	8.4	8.4	91.6	40.8	40.8	59.2
11.2	23.0	45.1	54.9	20.2	28.6	71.4	21.1	61.8	38.2
5.6	28.0	73.0	27.0	34.9	63.5	36.5	21.2	83.0	17.0
2.8	15.4	88.4	11.6	21.1	84.6	15.4	10.7	93.7	6.3
2	5.1	93.5	6.5	6.9	91.5	8.5	3.3	97.1	2.9
1.4	3.4	97.0	3.0	5.0	96.4	3.6	1.9	99.0	1.0
0.85	1.5	98.5	1.5	1.9	98.3	1.7	0.5	99.5	0.5
0.6	0.8	99.2	0.8	0.9	99.2	0.8	0.2	99.7	0.3
-0.6	0.8	-	-	0.8	-	-	0.3	-	-
Total	100.0	-	-	100.0	-	-	100.0	-	-

Table A1.2: +11.2mm Plant efficiency data

Feed % Ash : 41.6 Product % Ash : 20.6 Discard % Ash : 64.2 Product Yield : 51.8																	
Rel.Dens.	%Weight		% of Feed		Calc. Feed	Mean RD	Part. Coeff.	Fractional Ash			Cumulative Ash			Cumulative Yield			
	Prod	Disc	Prod	Disc				Prod	Disc	Feed	Prod	Disc	Feed	Prod	Disc	Feed	
F @ 1.30	0.77	0.00	0.40	0.00	0.40	1.30	100.0	12.0	0.0	12.0	12.0	0.0	12.0	0.4	0.0	0.4	
F @ 1.35	11.38	0.00	5.90	0.00	5.90	1.33	100.0	12.7	0.0	12.7	12.7	0.0	12.7	6.3	0.0	6.3	
F @ 1.40	20.87	0.00	10.82	0.00	10.82	1.38	100.0	15.5	0.0	15.5	14.5	0.0	14.5	17.1	0.0	17.1	
F @ 1.45	26.63	0.00	13.81	0.00	13.81	1.43	100.0	20.4	0.0	20.4	17.1	0.0	17.1	30.9	0.0	30.9	
F @ 1.50	17.71	0.00	9.18	0.00	9.18	1.48	100.0	24.4	0.0	24.4	18.8	0.0	18.8	40.1	0.0	40.1	
F @ 1.55	9.77	0.97	5.06	0.47	5.53	1.53	91.5	28.2	21.2	27.6	19.8	21.2	19.8	45.2	0.5	45.6	
F @ 1.60	12.86	0.85	6.67	0.41	7.07	1.58	94.2	31.6	27.6	31.4	21.3	24.2	21.4	51.8	0.9	52.7	
F @ 1.65	0.00	3.47	0.00	1.67	1.67	1.63	0.0	0.0	37.1	37.1	0.0	32.7	21.9	51.8	2.5	54.4	
F @ 1.70	0.00	3.43	0.00	1.65	1.65	1.68	0.0	0.0	41.1	41.1	0.0	36.0	22.4	51.8	4.2	56.0	
F @ 1.75	0.00	2.28	0.00	1.10	1.10	1.73	0.0	0.0	42.5	42.5	0.0	37.3	22.8	51.8	5.3	57.1	
F @ 1.80	0.00	4.44	0.00	2.14	2.14	1.78	0.0	0.0	47.7	47.7	0.0	40.3	23.7	51.8	7.4	59.3	
S @ 1.80	0.00	84.56	0.00	40.73	40.73	1.80	0.0	0.0	75.5	75.5	0.0	70.1	44.8	51.8	48.2	100.0	
Whole Coal	100.00	100.00	51.83	48.17	100.00												

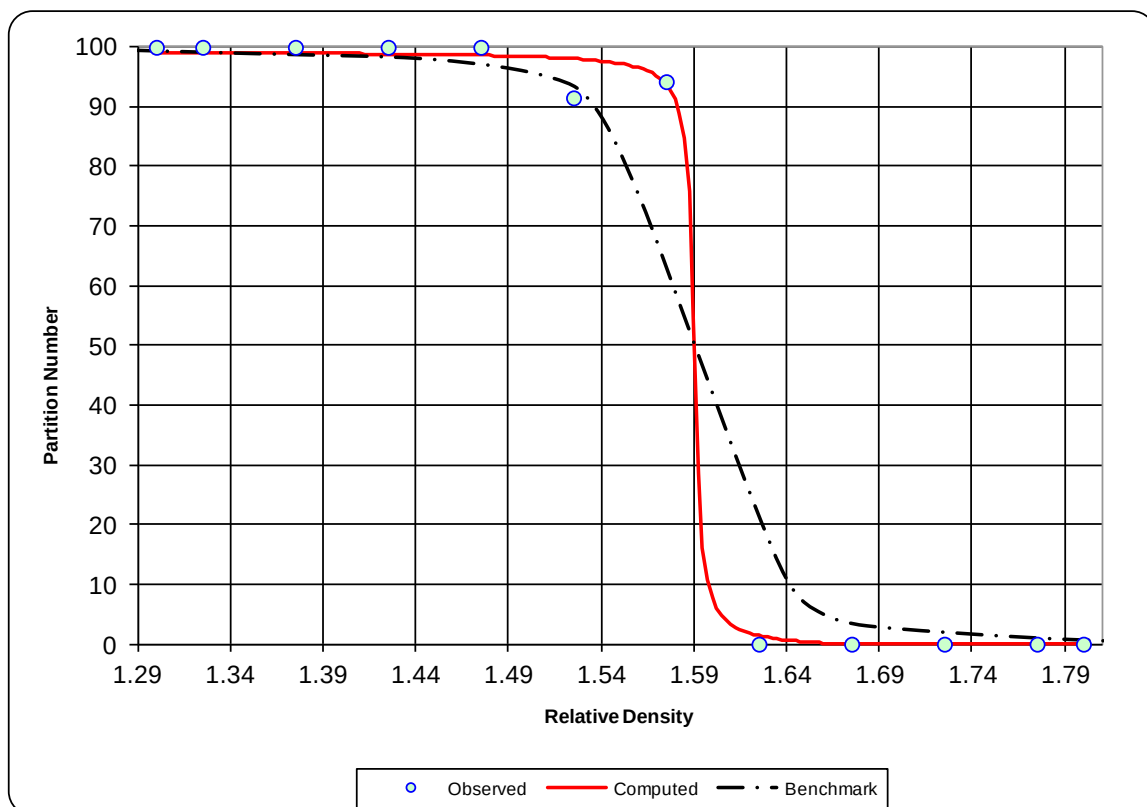


Figure A1.1: +11.2mm Plant efficiency partition curve

Table A1.3: +5.6mm Plant efficiency data

Feed % Ash : 29.1 Product % Ash : 19.4 Discard % Ash : 54.7 Product Yield : 72.5																	
Rel.Dens.	% Weight		% of Feed		Calc. Feed	Mean RD	Part. Coeff.	Fractional Ash			Cumulative Ash			Cumulative Yield			
	Prod	Disc	Prod	Disc				Prod	Disc	Feed	Prod	Disc	Feed	Prod	Disc	Feed	
F @ 1.30	3.99	0.00	2.89	0.00	2.89	1.30	100.0	7.1	0.0	7.1	7.1	0.0	7.1	2.9	0.0	2.9	
F @ 1.35	16.02	0.00	11.62	0.00	11.62	1.33	100.0	11.0	0.0	11.0	10.2	0.0	10.2	14.5	0.0	14.5	
F @ 1.40	25.19	0.00	18.27	0.00	18.27	1.38	100.0	15.2	0.0	15.2	13.0	0.0	13.0	32.8	0.0	32.8	
F @ 1.45	24.75	0.00	17.95	0.00	17.95	1.43	100.0	19.6	0.0	19.6	15.3	0.0	15.3	50.7	0.0	50.7	
F @ 1.50	14.15	0.00	10.26	0.00	10.26	1.48	100.0	24.4	0.0	24.4	16.9	0.0	16.9	61.0	0.0	61.0	
F @ 1.55	7.92	6.71	5.74	1.84	7.59	1.53	75.7	28.1	20.3	26.2	17.8	20.3	17.9	66.7	1.8	68.6	
F @ 1.60	7.98	1.98	5.79	0.54	6.33	1.58	91.4	32.3	27.3	31.9	19.0	21.9	19.1	72.5	2.4	74.9	
F @ 1.65	0.00	4.31	0.00	1.18	1.18	1.63	0.0	0.0	40.0	40.0	0.0	27.9	19.4	72.5	3.6	76.1	
F @ 1.70	0.00	6.14	0.00	1.69	1.69	1.68	0.0	0.0	42.6	42.6	0.0	32.6	19.9	72.5	5.3	77.8	
F @ 1.75	0.00	6.99	0.00	1.92	1.92	1.73	0.0	0.0	43.2	43.2	0.0	35.4	20.5	72.5	7.2	79.7	
F @ 1.80	0.00	8.05	0.00	2.21	2.21	1.78	0.0	0.0	46.5	46.5	0.0	38.1	21.2	72.5	9.4	81.9	
S @ 1.80	0.00	65.82	0.00	18.09	18.09	1.80	0.0	0.0	63.5	63.5	0.0	54.8	28.8	72.5	27.5	100.0	
Whole Coal	100.00	100.00	72.52	27.48	100.00												

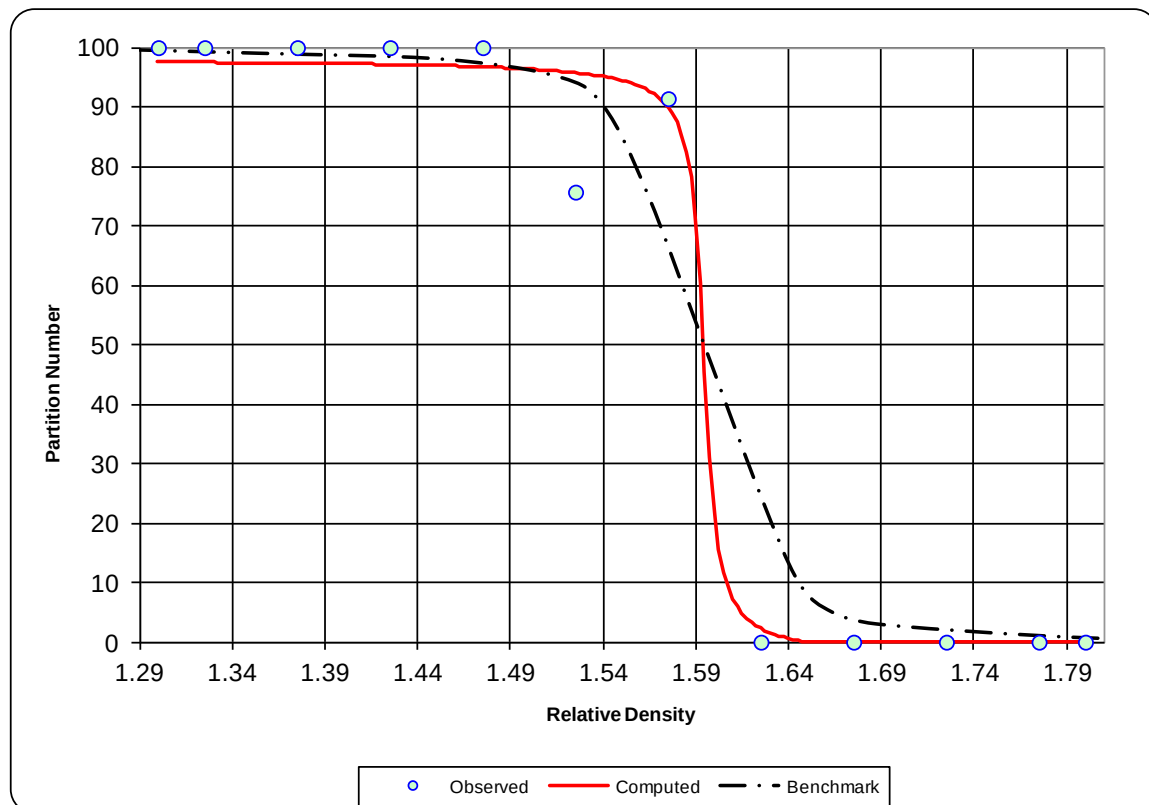
**Figure A1.2: +5.6mm Plant efficiency partition curve**

Table A1.4: +2.8mm Plant efficiency data

Feed % Ash : 21.5 Product % Ash : 17.3 Discard % Ash : 56.1 Product Yield : 89.3																	
Rel.Dens.	%Weight			% of Feed		Calc. Feed	Mean RD	Part. Coeff.	Fractional Ash			Cumulative Ash			Cumulative Yield		
	Prod	Disc		Prod	Disc				Prod	Disc	Feed	Prod	Disc	Feed	Prod	Disc	Feed
F @ 1.30	10.50	0.00		9.37	0.00	9.37	1.30	100.0	5.5	0.0	5.5	5.5	0.0	5.5	9.4	0.0	9.4
F @ 1.35	17.03	0.00		15.20	0.00	15.20	1.33	100.0	10.3	0.0	10.3	8.5	0.0	8.5	24.6	0.0	24.6
F @ 1.40	22.77	0.00		20.33	0.00	20.33	1.38	100.0	14.8	0.0	14.8	11.3	0.0	11.3	44.9	0.0	44.9
F @ 1.45	22.18	0.00		19.80	0.00	19.80	1.43	100.0	19.1	0.0	19.1	13.7	0.0	13.7	64.7	0.0	64.7
F @ 1.50	11.98	0.00		10.69	0.00	10.69	1.48	100.0	23.9	0.0	23.9	15.2	0.0	15.2	75.4	0.0	75.4
F @ 1.55	7.13	13.04		6.36	1.40	7.76	1.53	82.0	27.4	17.7	25.7	16.1	17.7	16.1	81.8	1.4	83.2
F @ 1.60	8.42	2.37		7.51	0.25	7.77	1.58	96.7	32.6	25.8	32.4	17.5	18.9	17.5	89.3	1.7	90.9
F @ 1.65	0.00	2.64		0.00	0.28	0.28	1.63	0.0	0.0	46.3	46.3	0.0	22.9	17.6	89.3	1.9	91.2
F @ 1.70	0.00	4.48		0.00	0.48	0.48	1.68	0.0	0.0	46.4	46.4	0.0	27.6	17.8	89.3	2.4	91.7
F @ 1.75	0.00	9.62		0.00	1.03	1.03	1.73	0.0	0.0	50.7	50.7	0.0	34.5	18.1	89.3	3.5	92.7
F @ 1.80	0.00	10.54		0.00	1.13	1.13	1.78	0.0	0.0	54.2	54.2	0.0	39.4	18.6	89.3	4.6	93.8
S @ 1.80	0.00	57.31		0.00	6.15	6.15	1.80	0.0	0.0	65.2	65.2	0.0	54.2	21.4	89.3	10.7	100.0
Whole Coal	100.00	100.00		89.26	10.74	100.00											

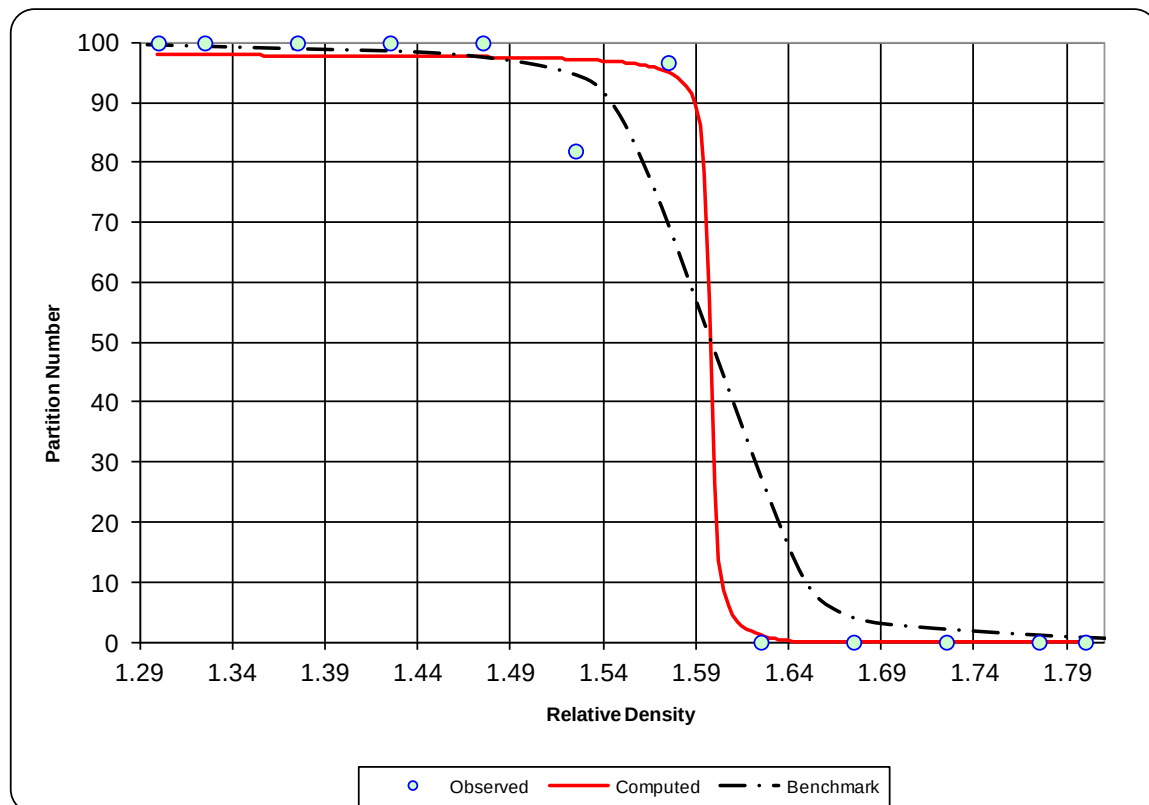


Figure A1.3: +2.8mm Plant efficiency partition curve

Table A1.5: +1.4mm Plant efficiency data

Feed % Ash : 23.2 Product % Ash : 14.9 Discard % Ash : 52.7 Product Yield : 78.0																	
Rel.Dens.	%Weight			% of Feed		Calc. Feed	Mean RD	Part. Coeff.	Fractional Ash			Cumulative Ash			Cumulative Yield		
	Prod	Disc		Prod	Disc				Prod	Disc	Feed	Prod	Disc	Feed	Prod	Disc	Feed
F @ 1.30	21.79	0.00		17.01	0.00	17.01	1.30	100.0	4.2	0.0	4.2	4.2	0.0	4.2	17.0	0.0	17.0
F @ 1.35	19.68	0.00		15.36	0.00	15.36	1.33	100.0	9.1	0.0	9.1	6.5	0.0	6.5	32.4	0.0	32.4
F @ 1.40	19.68	0.00		15.36	0.00	15.36	1.38	100.0	13.8	0.0	13.8	8.9	0.0	8.9	47.7	0.0	47.7
F @ 1.45	16.17	0.00		12.62	0.00	12.62	1.43	100.0	18.3	0.0	18.3	10.8	0.0	10.8	60.3	0.0	60.3
F @ 1.50	9.14	0.00		7.13	0.00	7.13	1.48	100.0	22.6	0.0	22.6	12.1	0.0	12.1	67.5	0.0	67.5
F @ 1.55	5.45	17.91		4.25	3.93	8.18	1.53	52.0	26.5	15.5	21.2	12.9	15.5	13.1	71.7	3.9	75.7
F @ 1.60	4.75	3.86		3.70	0.85	4.55	1.58	81.4	30.1	20.0	28.2	13.8	16.3	13.9	75.4	4.8	80.2
F @ 1.65	3.34	6.61		2.61	1.45	4.06	1.63	64.2	35.6	48.8	40.3	14.5	23.9	15.2	78.0	6.2	84.3
F @ 1.70	0.00	7.44		0.00	1.63	1.63	1.68	0.0	0.0	51.7	51.7	0.0	29.7	15.9	78.0	7.9	85.9
F @ 1.75	0.00	3.86		0.00	0.85	0.85	1.73	0.0	0.0	53.8	53.8	0.0	32.0	16.3	78.0	8.7	86.8
F @ 1.80	0.00	4.96		0.00	1.09	1.09	1.78	0.0	0.0	58.9	58.9	0.0	35.0	16.8	78.0	9.8	87.8
S @ 1.80	0.00	55.37		0.00	12.16	12.16	1.80	0.0	0.0	64.6	64.6	0.0	51.4	22.6	78.0	22.0	100.0
Whole Coal	100.00	100.00		78.04	21.96	100.00											

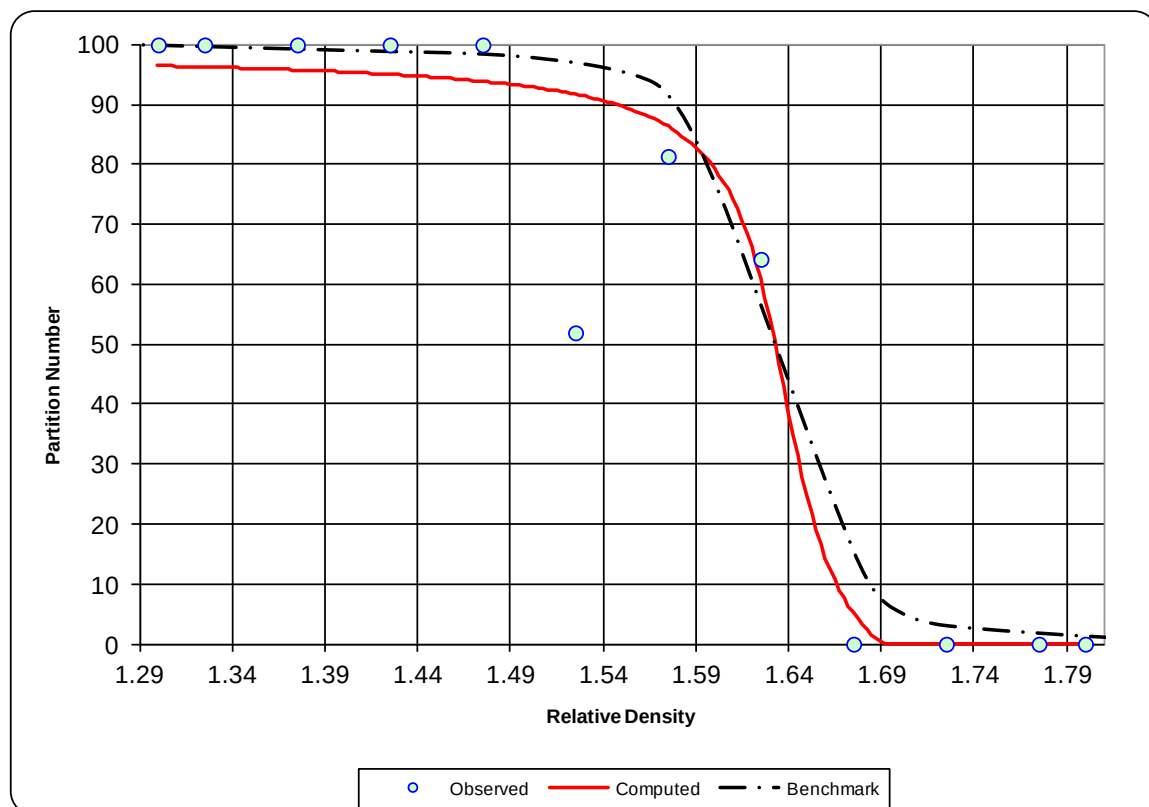
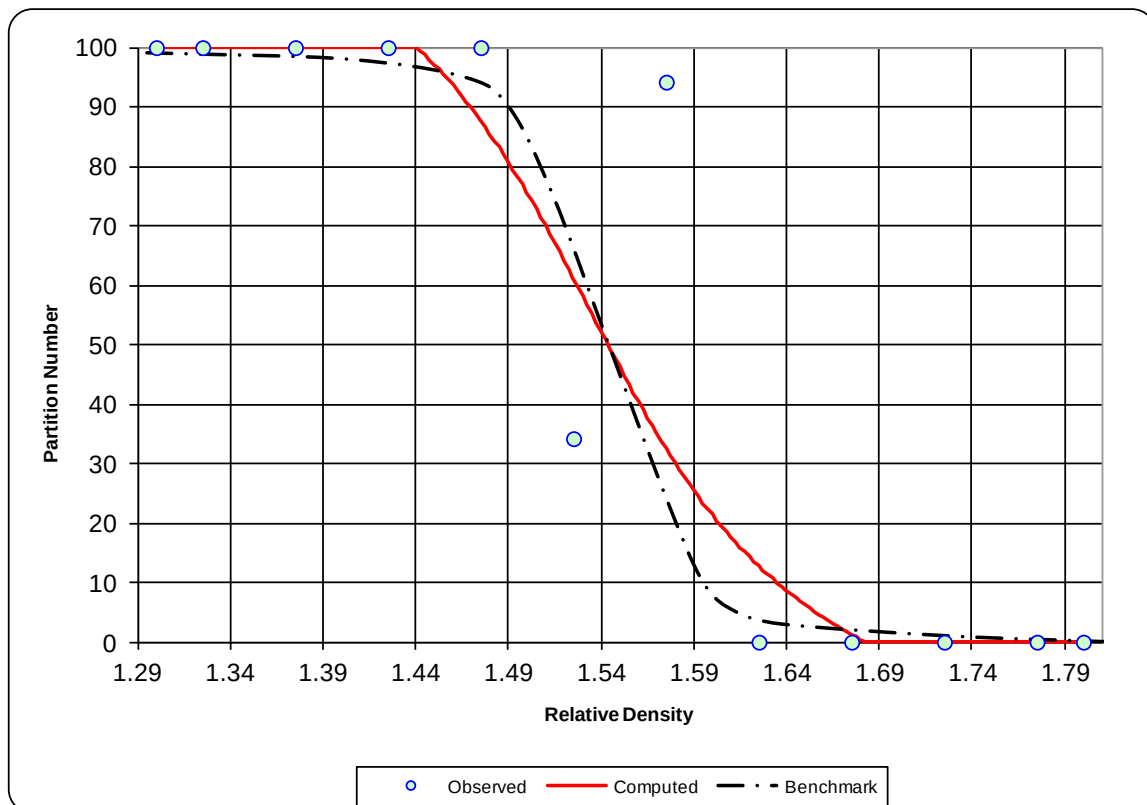
**Figure A1.4: +1.4mm Plant efficiency partition curve**

Table A1.6: +0.6mm Plant efficiency data

Feed % Ash : 20.5 Product % Ash : 13.8 Discard % Ash : 47.3 Product Yield : 80.0																	
Rel.Dens.	%Weight			% of Feed		Calc. Feed	Mean RD	Part. Coeff.	Fractional Ash			Cumulative Ash			Cumulative Yield		
	Prod	Disc		Prod	Disc				Prod	Disc	Feed	Prod	Disc	Feed	Prod	Disc	Feed
F @ 1.30	35.29	0.00		28.24	0.00	28.24	1.30	100.0	4.0	0.0	4.0	4.0	0.0	4.0	28.2	0.0	28.2
F @ 1.35	19.12	0.00		15.29	0.00	15.29	1.33	100.0	9.1	0.0	9.1	5.8	0.0	5.8	43.5	0.0	43.5
F @ 1.40	14.71	0.00		11.76	0.00	11.76	1.38	100.0	14.3	0.0	14.3	7.6	0.0	7.6	55.3	0.0	55.3
F @ 1.45	11.76	0.00		9.41	0.00	9.41	1.43	100.0	19.4	0.0	19.4	9.3	0.0	9.3	64.7	0.0	64.7
F @ 1.50	6.62	0.00		5.29	0.00	5.29	1.48	100.0	25.3	0.0	25.3	10.5	0.0	10.5	70.0	0.0	70.0
F @ 1.55	3.68	28.26		2.94	5.65	8.59	1.53	34.2	32.7	13.9	20.3	11.4	13.9	11.6	72.9	5.7	78.6
F @ 1.60	8.82	2.17		7.06	0.43	7.49	1.58	94.2	39.2	22.7	38.2	13.9	14.5	13.9	80.0	6.1	86.1
F @ 1.65	0.00	4.35		0.00	0.87	0.87	1.63	0.0	0.0	49.2	49.2	0.0	18.9	14.3	80.0	7.0	87.0
F @ 1.70	0.00	6.52		0.00	1.30	1.30	1.68	0.0	0.0	55.6	55.6	0.0	24.7	14.9	80.0	8.3	88.3
F @ 1.75	0.00	2.17		0.00	0.43	0.43	1.73	0.0	0.0	52.1	52.1	0.0	26.0	15.1	80.0	8.7	88.7
F @ 1.80	0.00	2.17		0.00	0.43	0.43	1.78	0.0	0.0	53.4	53.4	0.0	27.3	15.3	80.0	9.1	89.1
S @ 1.80	0.00	54.35		0.00	10.87	10.87	1.80	0.0	0.0	62.0	62.0	0.0	46.2	20.3	80.0	20.0	100.0
Whole Coal	100.00	100.00		80.00	20.00	100.00											

**Figure A1.5: +0.6mm Plant efficiency partition curve**

Appendix 1: Quantifying the Losses

Table A1.7: Overall Plant efficiency data

Feed % Ash : 34.1 Product % Ash : 18.5 Discard % Ash : 64.0 Product Yield : 65.7																	
Rel.Dens.	% Weight		% of Feed		Calc. Feed	Mean RD	Part. Coeff.	Fractional Ash			Cumulative Ash			Cumulative Yield			
	Prod	Disc	Prod	Disc				Prod	Disc	Feed	Prod	Disc	Feed	Prod	Disc	Feed	
F @ 1.30	7.45	0.00	4.89	0.00	4.89	1.30	100.0	5.3	0.0	5.3	5.3	0.0	5.3	4.9	0.0	4.9	
F @ 1.35	15.42	0.00	10.14	0.00	10.14	1.33	100.0	10.8	0.0	10.8	9.1	0.0	9.1	15.0	0.0	15.0	
F @ 1.40	22.48	0.00	14.77	0.00	14.77	1.38	100.0	15.0	0.0	15.0	12.0	0.0	12.0	29.8	0.0	29.8	
F @ 1.45	23.36	0.00	15.35	0.00	15.35	1.43	100.0	19.7	0.0	19.7	14.6	0.0	14.6	45.2	0.0	45.2	
F @ 1.50	13.91	0.00	9.14	0.00	9.14	1.48	100.0	24.2	0.0	24.2	16.2	0.0	16.2	54.3	0.0	54.3	
F @ 1.55	7.87	4.57	5.17	1.57	6.74	1.53	76.7	27.9	18.4	25.7	17.2	18.4	17.3	59.5	1.6	61.0	
F @ 1.60	9.12	1.42	5.99	0.49	6.48	1.58	92.5	32.1	26.0	31.7	18.6	20.2	18.7	65.5	2.1	67.5	
F @ 1.65	0.40	3.73	0.26	1.28	1.54	1.63	17.0	0.0	39.7	32.9	0.0	27.7	19.0	65.7	3.3	69.0	
F @ 1.70	0.00	4.35	0.00	1.49	1.49	1.68	0.0	0.0	43.2	43.2	0.0	32.5	19.5	65.7	4.8	70.5	
F @ 1.75	0.00	4.16	0.00	1.43	1.43	1.73	0.0	0.0	45.4	45.4	0.0	35.4	20.0	65.7	6.2	72.0	
F @ 1.80	0.00	5.88	0.00	2.02	2.02	1.78	0.0	0.0	49.1	49.1	0.0	38.8	20.8	65.7	8.3	74.0	
S @ 1.80	0.00	75.90	0.00	26.02	26.02	1.80	0.0	0.0	72.0	72.0	0.0	64.0	34.1	65.7	34.3	100.0	
Whole Coal	100.00	100.00	65.71	34.29	100.00												

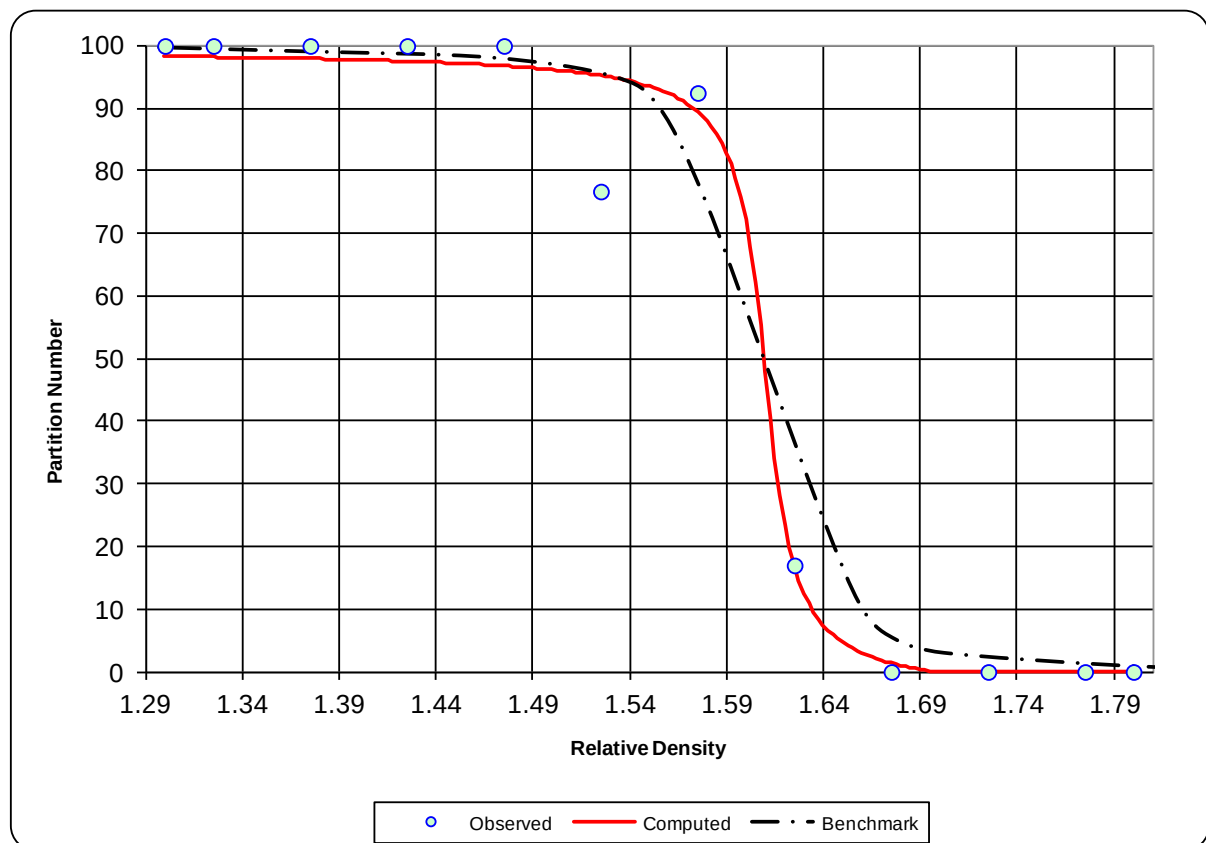


Figure A1.6: Overall Plant efficiency partition curve

A1.2 Examination of the effect of particle size in the flotation circuit

A1.2.1 Mutale Shaft

Table A1.8: Mutale flotation performance by size fraction

Date	Size	Ash (%)			Yield (%)	Fractional Sizing (%)			Cumulative Passing Sizing (%)		
		Feed	Product	Tailings		Feed	Product	Tailings	Feed	Product	Tailings
23/07/2008	+1.4mm	36.3	9.0	37.5	4.2	12.8	0.7	56.1	100.0	100.0	100.0
	-1.4+1.0mm	32.5	6.3	33.0	1.9	13.3	4.9	8.8	87.2	99.3	43.9
	-1.0+0.5mm	24.6	7.0	40.5	47.5	20.5	30.9	17.5	73.9	94.3	35.1
	-0.5+0.3mm	21.8	12.1	56.1	78.0	14.3	31.7	6.3	53.4	63.4	17.6
	-0.3+0.212mm	24.3	16.6	68.2	85.1	7.5	22.5	2.2	39.2	31.7	11.2
	-0.212mm	18.6	18.3	68.7	99.4	31.7	9.2	9.0	31.7	9.2	9.0
15/08/2008	+1.4mm	26.4	7.9	29.8	15.6	19.5	4.3	31.4	100.0	100.0	100.0
	-1.4+1.0mm	24.0	6.9	37.5	44.0	20.3	10.7	25.9	80.5	95.7	68.6
	-1.0+0.5mm	23.2	8.7	47.8	62.7	29.4	25.8	26.5	60.2	85.0	42.8
	-0.5+0.3mm	20.3	11.0	70.4	84.2	18.1	23.7	9.0	30.8	59.2	16.3
	-0.3+0.212mm	24.9	12.3	80.8	81.6	7.6	16.1	3.5	12.7	35.5	7.3
	-0.212mm	42.7	16.7	84.1	61.5	5.1	19.5	3.9	5.1	19.5	3.9
20/08/2008	+1.4mm	29.2	7.0	34.1	18.2	15.4	2.8	42.1	100.0	100.0	100.0
	-1.4+1.0mm	24.4	6.3	37.0	41.0	16.9	8.1	24.0	84.6	97.2	57.9
	-1.0+0.5mm	20.1	8.0	49.3	70.6	30.7	23.6	22.8	67.7	89.1	33.9
	-0.5+0.3mm	18.1	10.9	67.3	87.1	14.5	23.2	6.4	37.1	65.5	11.1
	-0.3+0.212mm	21.5	12.9	79.3	87.1	8.2	15.3	2.5	22.6	42.3	4.7
	-0.212mm	25.9	14.9	79.0	82.9	14.4	27.0	2.2	14.4	27.0	2.2

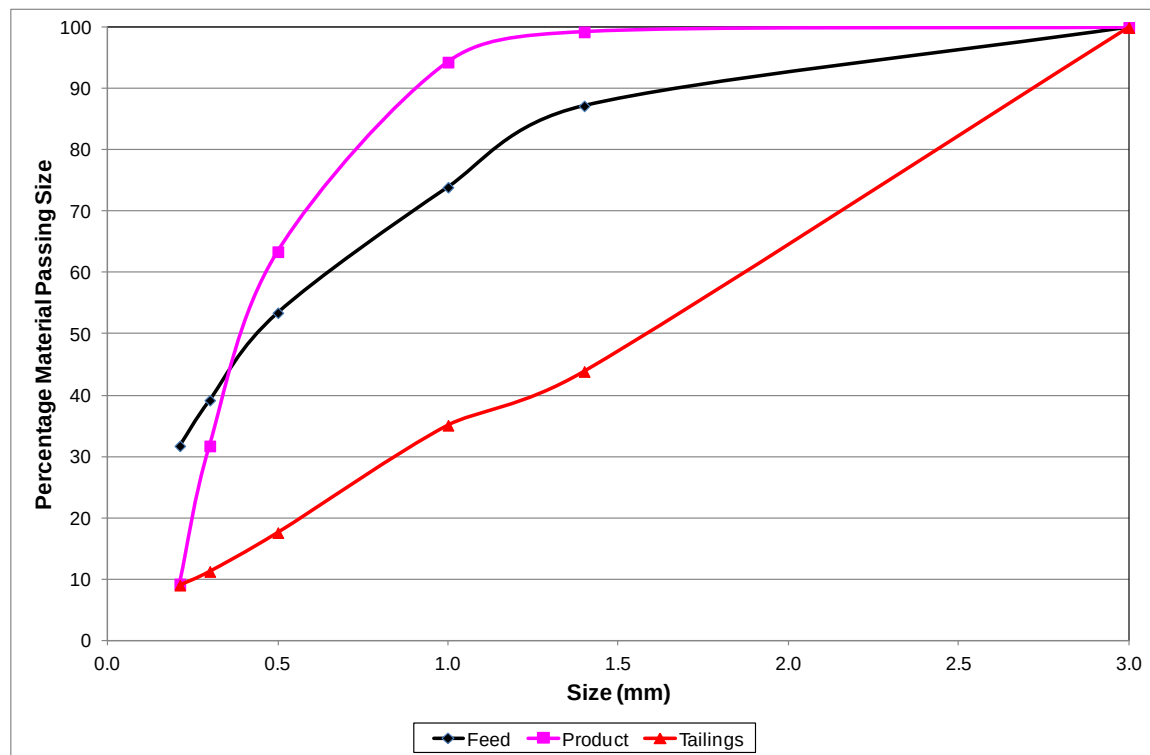


Figure A1.7: Mutale Sample 23/07/2008

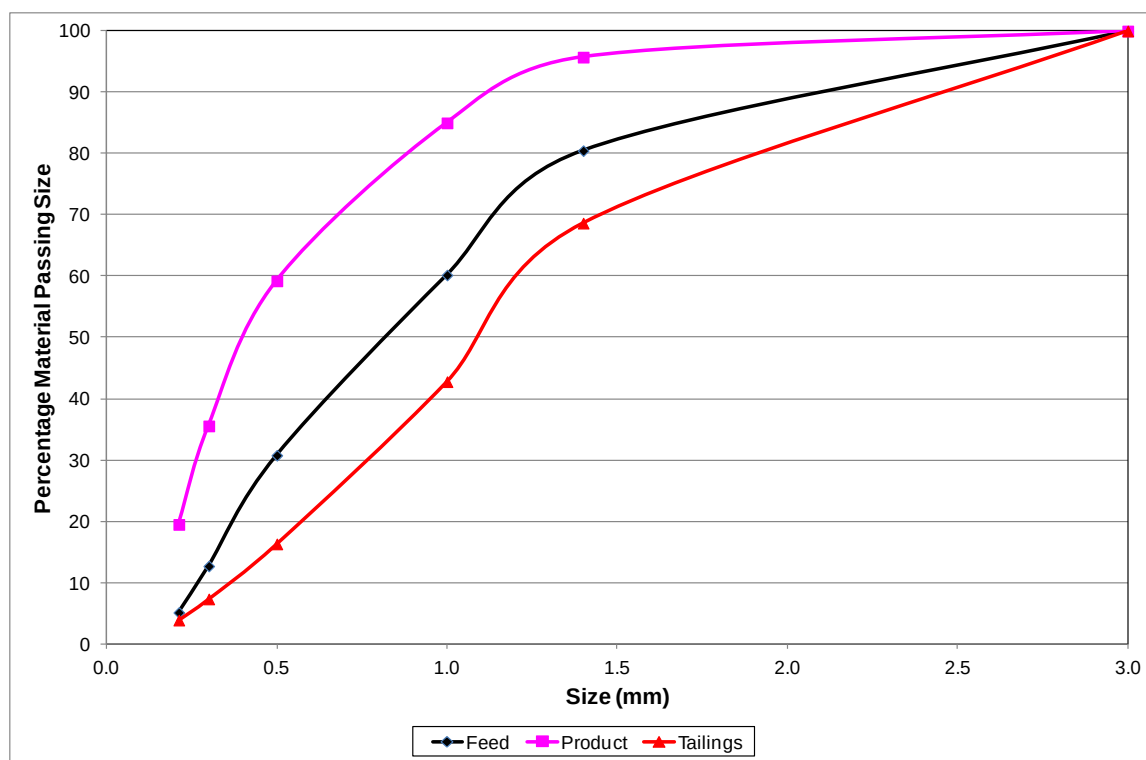


Figure A1.8: Mutale Sample 15/08/2008

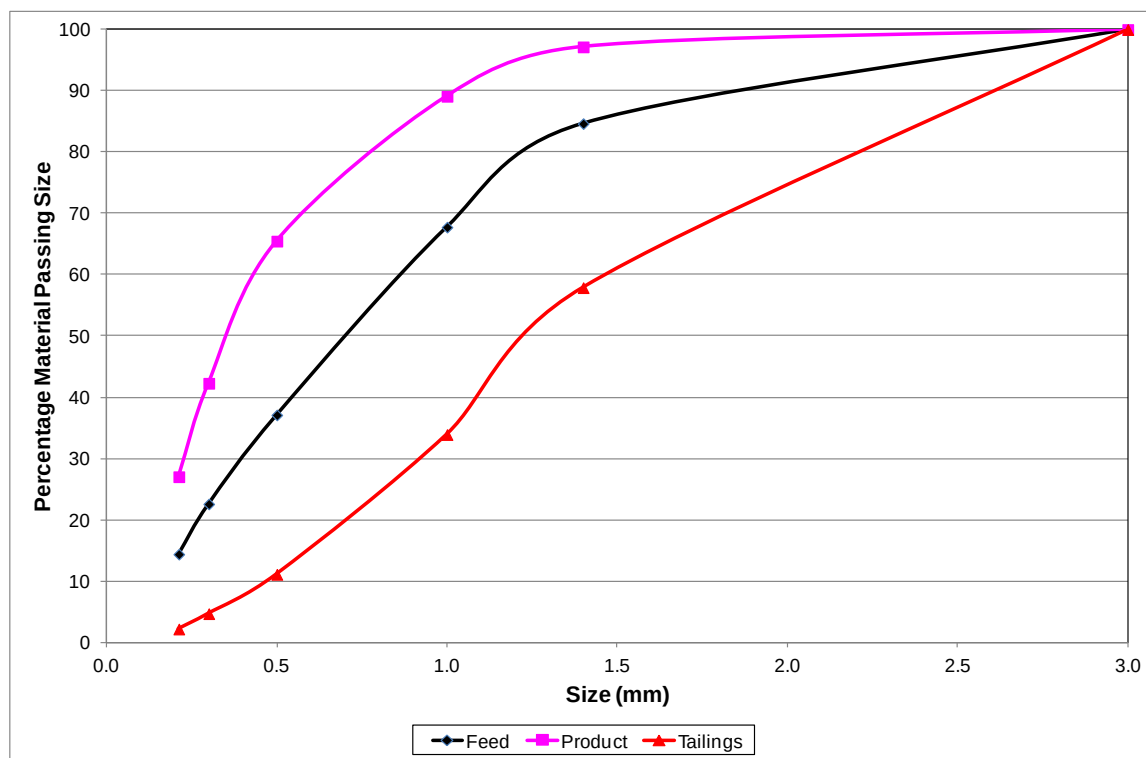


Figure A1.9: Mutale Sample 20/08/2008

A1.2.1 Vhukati Shaft

Table A1.9: Vhukati flotation performance by size fraction

Date	Size	Ash (%)			Yield (%)	Fractional Sizing (%)			Cumulative Passing Sizing (%)		
		Feed	Product	Tailings		Feed	Product	Tailings	Feed	Product	Tailings
28/07/2008	+1.4mm	27.6	6.8	34.3	24.4	15.6	1.7	71.6	100.0	100.0	100.0
	-1.4+1.0mm	23.7	7.0	30.2	28.0	23.7	9.9	6.9	84.4	98.3	28.4
	-1.0+0.5mm	14.6	8.3	45.0	82.8	26.7	32.5	10.6	60.7	88.4	21.5
	-0.5+0.3mm	14.8	11.3	61.7	93.1	17.2	26.0	3.8	33.9	55.9	10.9
	-0.3+0.212mm	14.1	13.4	68.6	98.7	4.1	9.9	2.4	16.7	30.0	7.1
	-0.212mm	17.5	11.3	73.9	90.1	12.6	20.0	4.6	12.6	20.0	4.6
12/08/2008	+1.4mm	25.7	8.4	26.9	6.3	20.2	3.3	53.7	100.0	100.0	100.0
	-1.4+1.0mm	21.7	7.6	31.2	40.2	20.2	4.9	19.8	79.8	96.7	46.3
	-1.0+0.5mm	22.8	8.9	43.0	59.1	33.8	29.9	20.6	59.6	91.8	26.4
	-0.5+0.3mm	22.0	11.0	26.9	31.2	15.0	22.8	3.8	25.9	61.9	5.8
	-0.3+0.212mm	21.3	13.3	66.3	85.0	6.0	17.2	0.9	10.9	39.0	2.0
	-0.212mm	34.3	17.7	74.6	70.9	4.9	21.8	1.1	4.9	21.8	1.1
21/08/2008	+1.4mm	26.0	6.8	29.8	16.6	21.1	4.1	36.7	100.0	100.0	100.0
	-1.4+1.0mm	21.7	6.7	31.3	39.1	17.0	9.0	23.6	78.9	95.9	63.3
	-1.0+0.5mm	21.1	8.3	45.3	65.4	30.7	25.5	25.2	61.9	86.9	39.6
	-0.5+0.3mm	20.4	10.2	62.3	80.5	14.9	25.8	8.0	31.3	61.4	14.5
	-0.3+0.212mm	24.5	11.9	71.5	78.9	5.8	16.6	3.0	16.4	35.6	6.5
	-0.212mm	34.6	15.8	77.4	69.4	10.6	19.0	3.5	10.6	19.0	3.5

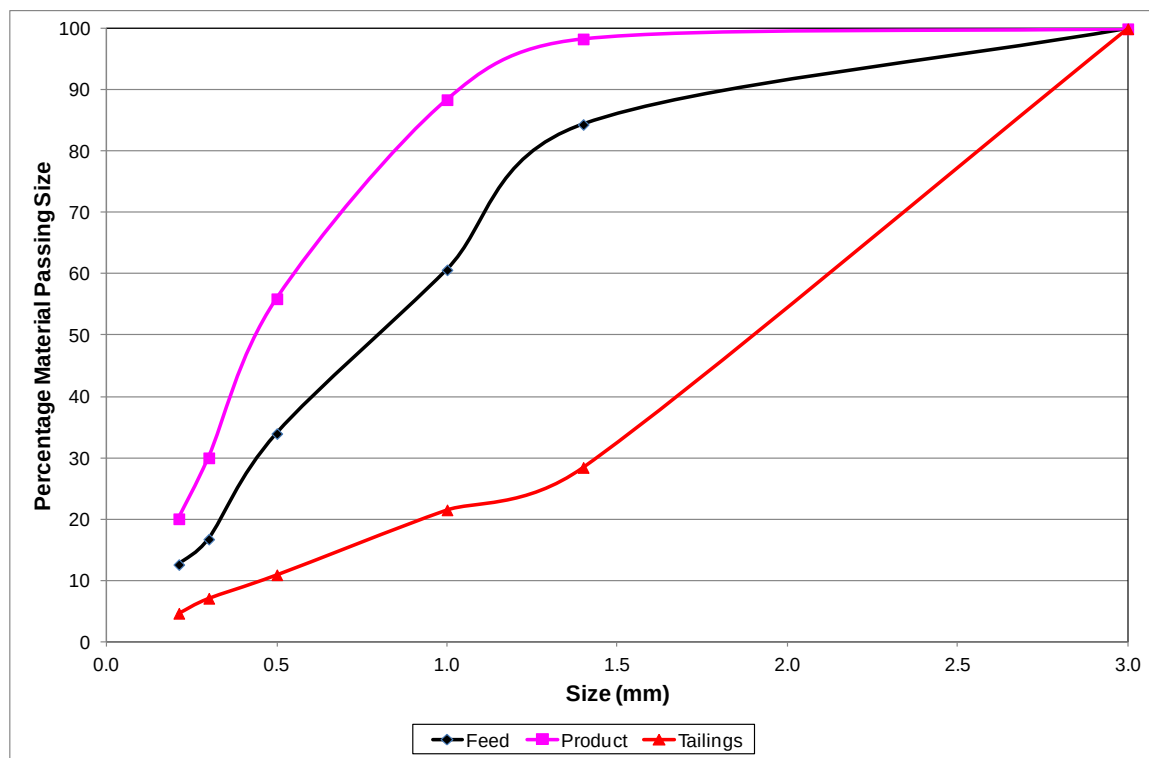


Figure A1.10: Vhukati Sample 28/07/2008

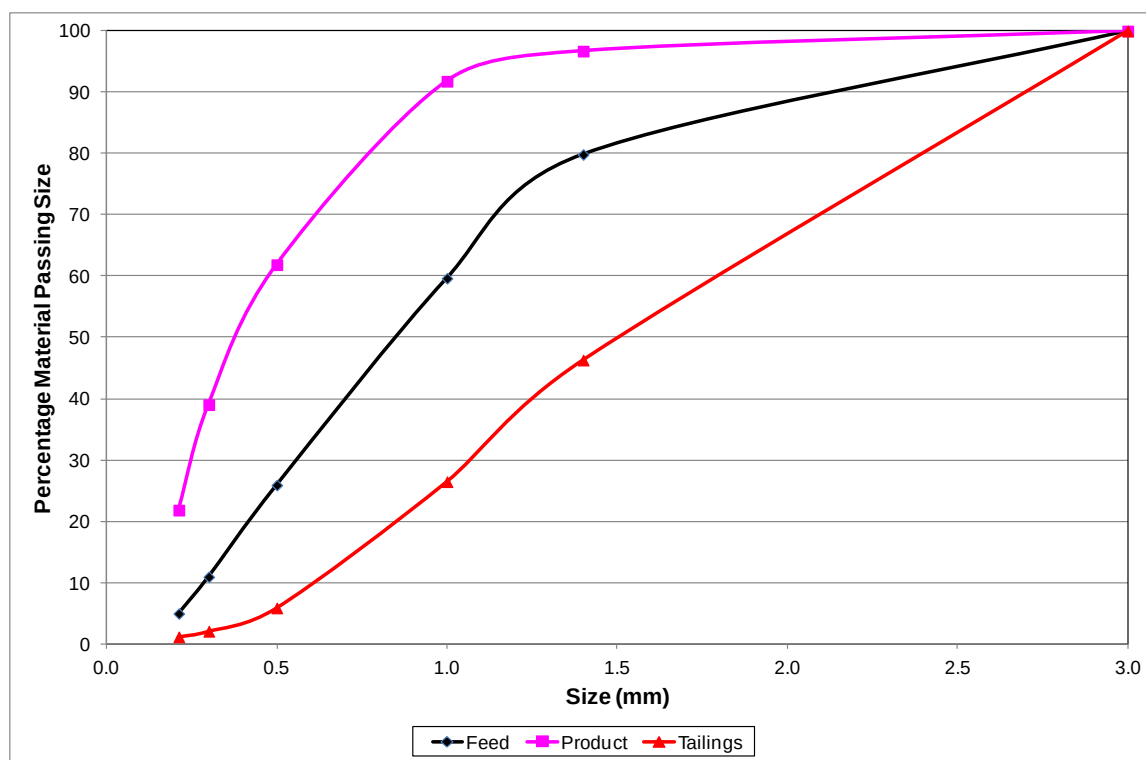


Figure A1.11: Vhukati Sample 12/08/2008

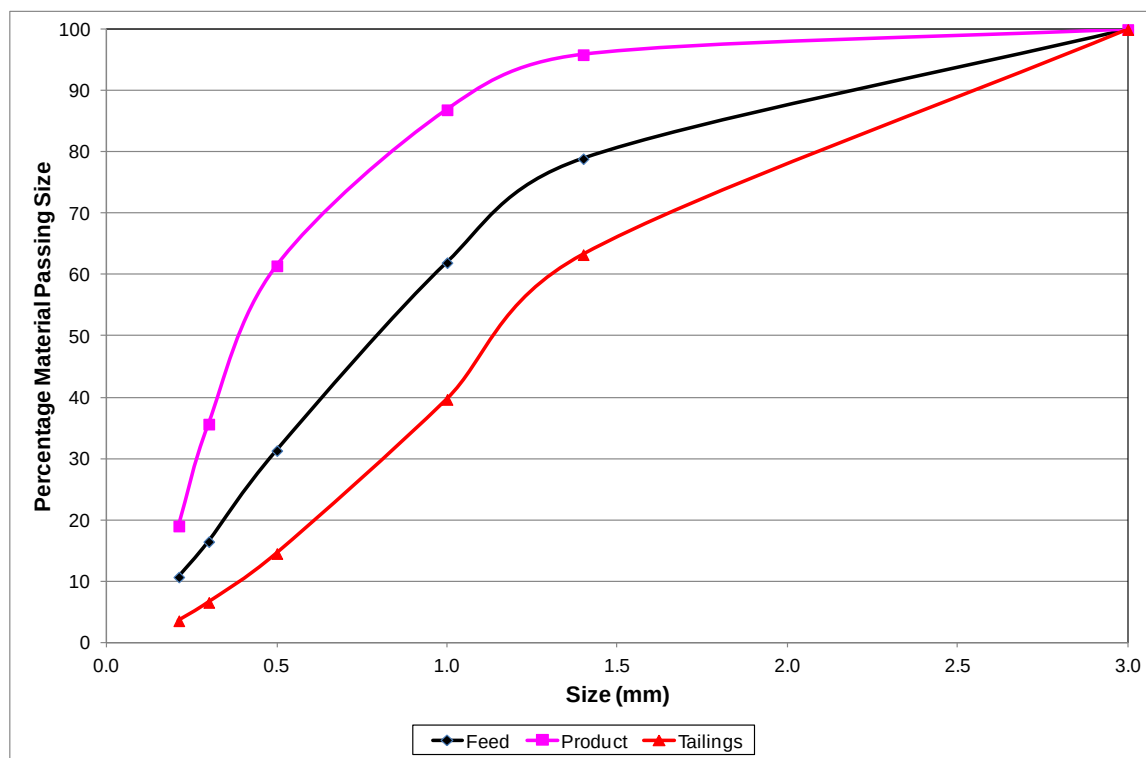


Figure A1.12: Vhukati Sample 21/08/2008

A1.2.3 Goni Shaft

Table A1.10: Goni flotation performance by size fraction

Date	Size	Fractional Sizing (%)			Cumulative Passing Sizing (%)			Ash (%)			Yield (%)
		Feed	Product	Tailings	Feed	Product	Tailings	Feed	Product	Tailings	
24/07/2008	+1.4mm	9.5	0.9	37.7	100.0	100.0	100.0	19.7	7.2	35.6	56.0
	-1.4+1.0mm	15.0	7.8	23.3	90.5	99.1	62.3	18	5.8	41.9	66.2
	-1.0+0.5mm	36.5	33.8	23.4	75.5	91.3	39.0	15.4	7.2	50.6	81.1
	-0.5+0.3mm	14.4	35.6	8.1	39.0	57.5	15.6	11.6	10.0	69.3	97.3
	-0.3+0.212mm	13.1	11.0	3.5	24.6	21.8	7.5	24.3	10.6	74.8	78.7
	-0.212mm	11.5	10.8	4.0	11.5	10.8	4.0	23.9	21.6	74.1	95.6
06/08/2008	+1.4mm	12.0	2.9	50.7	100.0	100.0	100.0	26.4	7.0	35.6	32.3
	-1.4+1.0mm	23.0	8.8	20.9	88.0	97.1	49.3	22.5	7.3	41.9	55.9
	-1.0+0.5mm	34.3	36.4	18.2	65.0	88.3	28.4	19.7	8.4	50.6	73.0
	-0.5+0.3mm	16.7	20.3	6.2	30.7	51.9	10.2	18.1	9.6	69.3	85.8
	-0.3+0.212mm	7.7	10.6	1.6	14.0	31.6	3.9	23.7	10.4	74.8	79.3
	-0.212mm	6.3	21.0	2.3	6.3	21.0	2.3	38.6	14.4	74.1	59.5
08/08/2008	+1.4mm	6.4	2.4	46.3	100.0	100.0	100.0	22.3	8.8	38.3	54.2
	-1.4+1.0mm	8.7	8.8	15.1	93.6	97.6	53.7	15.7	7.0	33.2	67.0
	-1.0+0.5mm	26.1	24.5	21.0	85.0	88.8	38.7	14.6	9.8	47.4	87.2
	-0.5+0.3mm	22.5	23.1	8.4	58.8	64.2	17.7	16.3	12.0	62.7	91.5
	-0.3+0.212mm	15.5	15.4	4.4	36.3	41.1	9.3	14.1	12.9	66.5	97.8
	-0.212mm	20.9	25.7	4.8	20.9	25.7	4.8	25.9	17.3	74.1	84.9
15 08 2008	+1.4mm	13.9	1.6	24.6	100.0	100.0	100.0	27.1	9.8	32.1	22.3
	-1.4+1.0mm	18.3	7.4	33.2	86.1	98.4	75.4	24.7	6.9	29.5	21.0
	-1.0+0.5mm	27.0	29.8	24.7	67.9	91.1	42.2	20.7	8.0	44.1	64.8
	-0.5+0.3mm	16.8	23.4	8.1	40.8	61.3	17.5	18.6	8.7	62.8	81.6
	-0.3+0.212mm	9.0	11.6	3.9	24.1	37.8	9.4	18.6	9.6	72.5	85.6
	-0.212mm	15.0	26.3	5.5	15.0	26.3	5.5	26.8	12.4	73.6	76.5

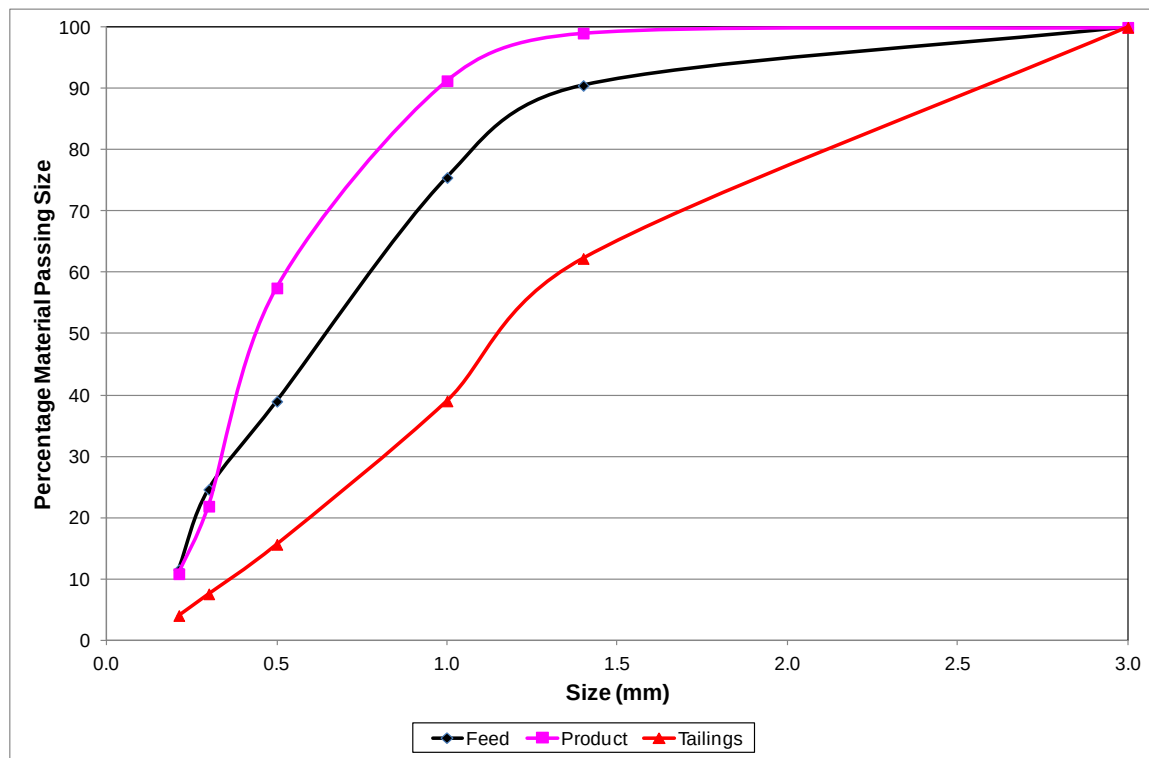


Figure A1.13: Goni Sample 24/07/2008

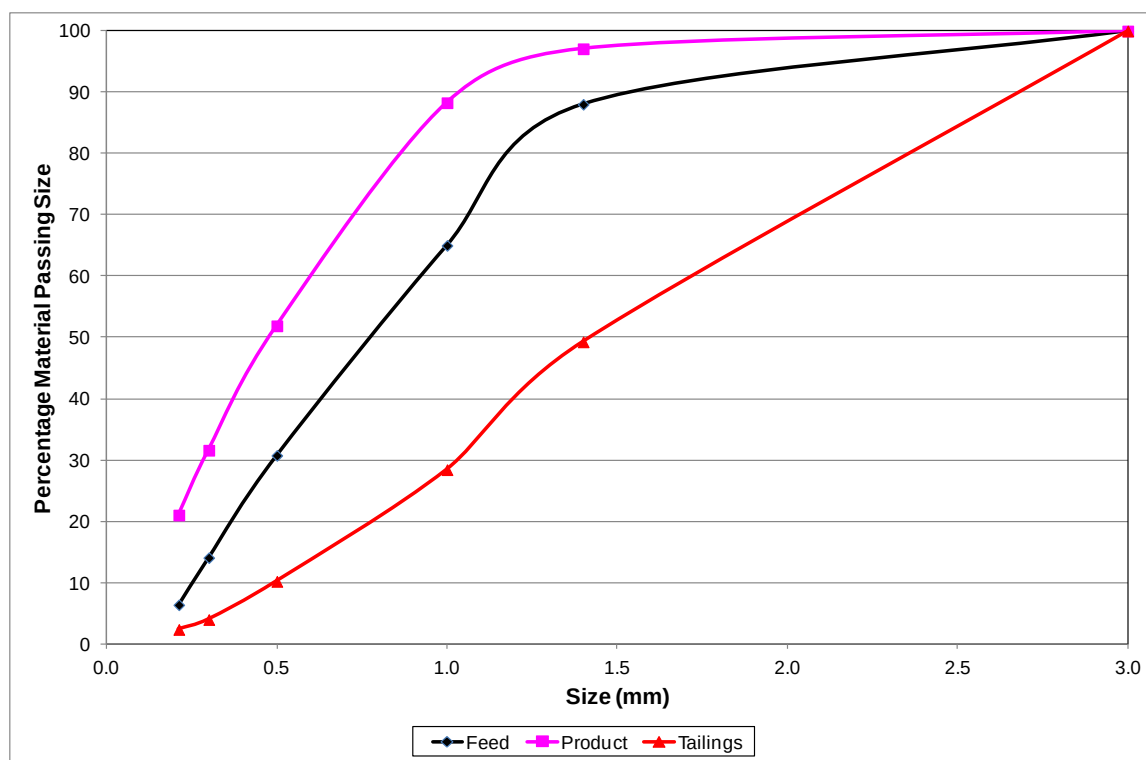


Figure A1.14: Goni Sample 06/08/2008

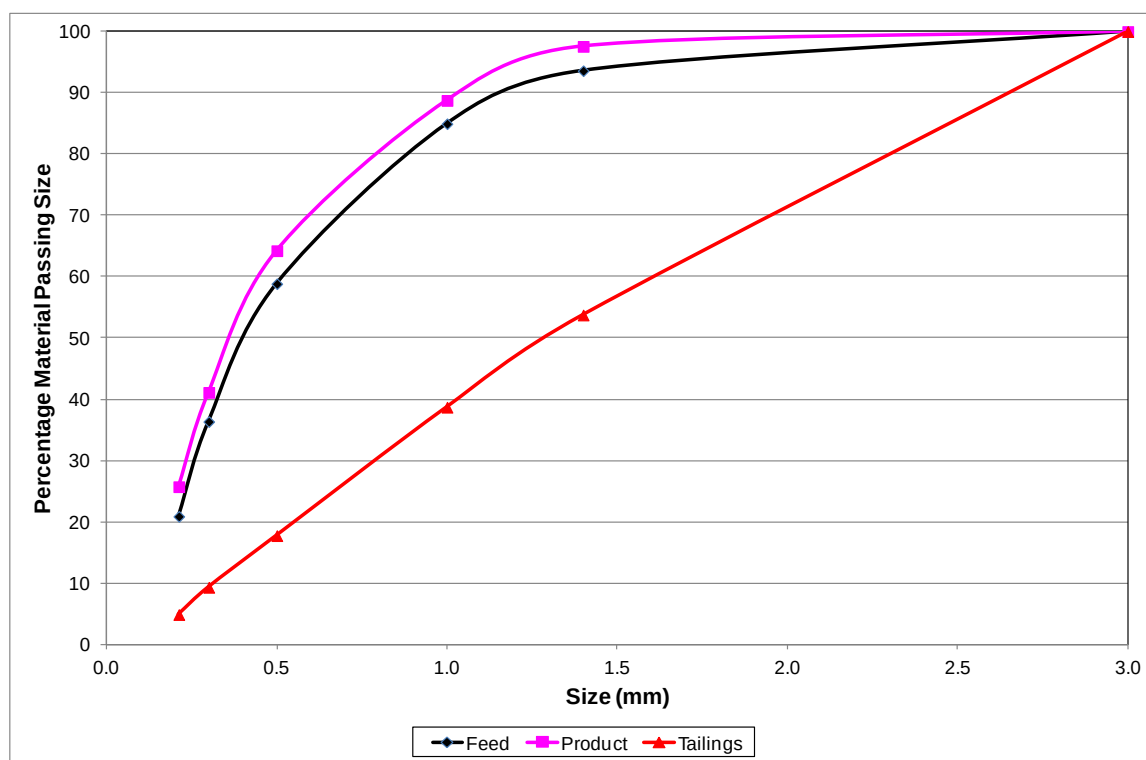


Figure A1.15: Goni Sample 08/08/2008

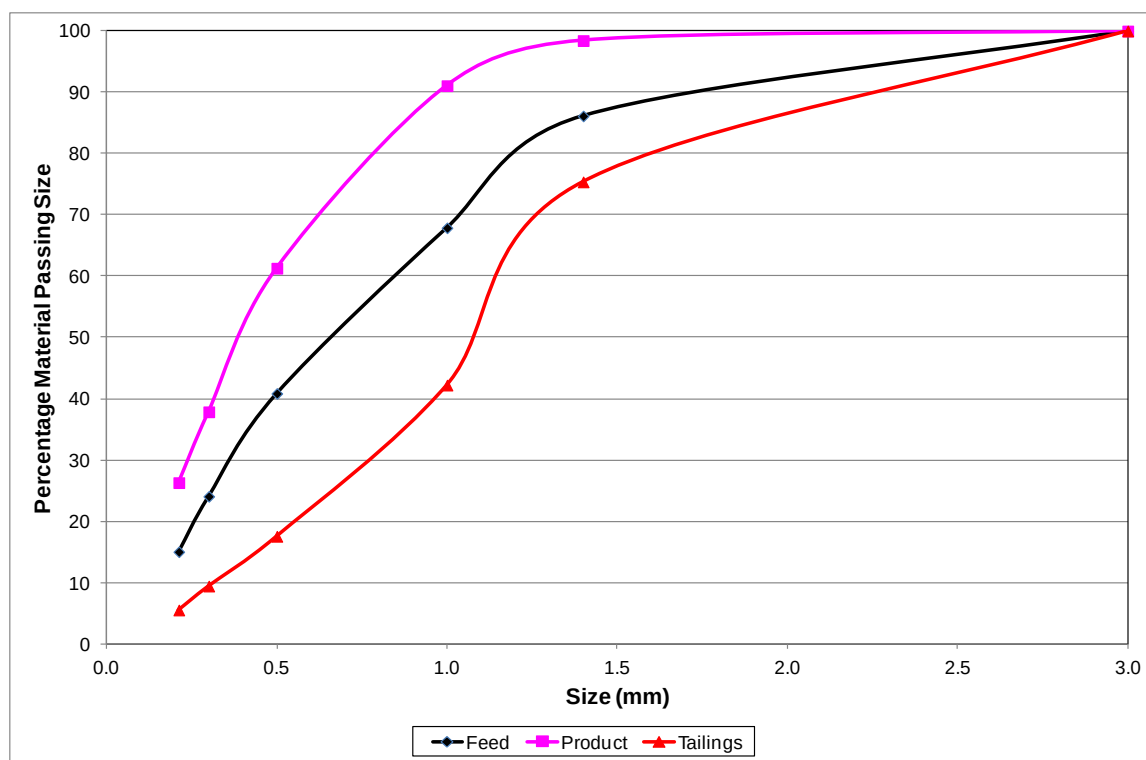


Figure A1.16: Goni Sample 15/08/2008

Appendix 2: Discard Dump Borehole Results

This appendix includes all the data related to the boreholes drilled during the auger campaign on both the old dump and the current dump.

The data includes the screening, destoning and flotation after milling. Flotation rate data is also included. The results presented here are discussed in Chapters 3, 4 and 6 of the thesis respectively depending upon the aspect under consideration.

A2.1 Old Dump Data

The results for the boreholes drilling on the old dump are presented in Table A2.1 (raw data) and Table A2.1 (adjusted data) below. During the initial processing and subsequent delivery to the Exxaro Research and Development in Pretoria, it was discovered that the -0.5mm material for two samples, namely section 3 of borehole 10 and section 1 of borehole 11, had been misplaced. The in-situ sample relative density and ash content were not affected by this and have been subjected to the subsequent descriptive statistics calculations. Although these samples were subjected to the destoning, milling and froth flotation test procedures, these results produced would not be representative of that particular area of the dump and have therefore also been excluded from the descriptive statistical calculations. The descriptive statistics are presented globally in Table A2.3 and by layer in Table A2.3 respectively.

It should also be noted that the two holes sampled from the slimes section were also excluded at this point due to their fineness and were examined separately.

Appendix 2: Discard Dump Borehole Results

Table A2.1: Old Dump data – raw data

Borehole	Section	Co-ordinates			Depth (m)		Insitu Results				+0.5mm Material		+0.5mm Floats RD1.80		+0.5mm Sinks RD1.80		-0.5mm Material		Flotation Feed		Flotation Product		Flotation Tailings		Final Tailings	
		X	Y	Z	From	To	Sample RD	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	+0.5mm S1.80 + Flotation Tailings		
BH1	1	5789.4	2490349.1	395.59	0	4.5	1.94	100.0	61.6	80.0	85.0	18.5	38.6	61.5	73.0	20.0	48.1	38.5	43.5	12.6	21.0	26.0	54.4	87.4	67.5	
	2				4.5	9.0	1.81	100.0	53.4	77.9	57.1	31.3	33.3	46.6	73.1	22.1	40.5	53.4	36.3	11.0	19.7	42.4	40.6	89.0	57.6	
	3				9	13.5	1.87	100.0	52.5	82.5	55.1	35.5	32.4	47.0	72.3	17.5	40.1	53.0	34.9	19.2	19.5	16.3	47.6	63.3	66.0	
	4				13.5	18.0	1.90	100.0	53.7	79.2	57.3	31.3	32.9	47.9	73.4	20.8	39.7	52.1	35.6	21.6	19.6	9.7	62.4	57.6	71.5	
BH2	1	5791.7	2490300.4	394.81	0	4.5	1.89	100.0	51.4	79.1	53.9	32.8	29.0	46.3	71.5	20.9	42.0	53.7	34.1	28.1	21.4	25.5	48.0	71.9	63.2	
	2				4.5	9.0	1.94	100.0	54.1	80.7	56.3	27.7	28.8	53.0	70.6	19.3	45.1	47.0	35.5	38.3	29.9	8.7	60.2	61.7	69.2	
	3				9	13.5	1.87	100.0	50.0	78.6	52.6	34.1	28.8	44.5	70.9	21.4	40.1	55.5	33.2	25.5	20.9	30.0	43.7	74.5	59.9	
	4				13.5	17.5	1.89	100.0	52.8	79.1	54.9	30.5	28.5	48.7	71.4	20.9	44.9	51.3	35.1	20.3	20.0	31.1	45.0	79.7	61.1	
BH3	1	5787.5	2490251.9	394.46	0	4.5	1.88	100.0	54.5	76.9	58.2	30.1	34.9	46.8	73.1	23.1	42.5	53.2	38.2	20.8	18.4	32.4	50.9	79.2	64.0	
	2				4.5	9.0	1.84	100.0	51.5	77.7	55.2	34.4	32.4	43.3	73.3	22.3	38.4	56.7	34.8	20.1	17.2	36.6	44.5	79.9	60.1	
	3				9	13.5	1.80	100.0	50.3	80.8	52.7	38.4	31.6	42.4	71.9	19.2	40.3	57.6	34.5	22.8	17.7	34.8	45.4	77.2	60.0	
	4				13.5	18.0	1.86	100.0	52.5	81.4	55.1	35.8	33.5	45.6	72.1	18.6	41.0	54.4	36.1	22.4	18.7	32.0	48.2	77.6	62.2	
BH4	1	5787.5	2490201.6	394.63	0	4.5	1.83	100.0	50.4	75.7	53.8	33.4	29.3	42.4	73.1	24.3	39.7	57.6	33.7	25.0	18.2	32.6	45.6	75.0	61.2	
	2				4.5	9.0	1.86	100.0	50.6	80.5	53.2	36.0	29.4	44.5	72.4	19.5	40.0	55.5	33.1	24.2	19.6	31.3	43.6	75.8	60.5	
	3				9	13.5	1.85	100.0	49.1	82.3	50.8	39.7	28.7	42.6	71.5	17.7	40.8	57.4	32.4	21.4	18.2	36.0	40.8	78.6	57.4	
	4				13.5	18.0	1.84	100.0	49.5	75.8	53.4	33.5	29.4	42.3	72.3	24.2	37.5	57.7	32.8	24.1	18.3	33.6	43.3	75.9	59.5	
BH5	1	5791.7	2490151.8	394.63	0	4.5	1.85	100.0	51.2	74.2	54.9	31.0	29.3	43.3	73.3	25.8	40.6	56.7	34.4	23.9	18.7	32.8	45.9	76.1	61.5	
	2				4.5	9.0	1.84	100.0	50.0	79.9	52.6	36.3	29.0	43.6	72.2	20.1	39.9	56.4	32.8	24.7	19.5	31.7	43.2	75.3	60.0	
	3				9	13.5	1.81	100.0	48.8	78.4	51.8	39.8	30.7	38.6	73.6	21.6	37.6	61.4	33.1	31.6	20.2	29.8	46.8	68.4	61.9	
	4				13.5	18.0	1.81	100.0	48.7	76.2	52.3	39.5	32.2	36.7	74.0	23.8	37.0	63.3	34.0	31.1	20.5	32.1	47.1	68.9	61.5	
BH6 (BH17a)	1	5681.6	2490075.9	392.19	0	4.5	1.81	100.0	52.9	80.7	55.6	33.5	29.3	42.2	73.8	19.3	42.0	52.8	33.9	20.5	18.0	32.3	44.1	74.5	60.9	
	2				4.5	9.0	1.90	100.0	51.0	82.8	53.0	35.8	28.9	40.4	73.6	17.2	41.1	53.0	32.9	21.8	18.0	31.3	43.2	71.7	60.3	
	3				9	13.5	1.88	100.0	49.2	79.6	51.6	37.0	28.1	41.1	73.8	20.4	39.9	57.4	32.3	22.9	17.2	34.5	42.3	75.6	59.5	
	4				13.5	18.0	1.88	100.0	50.5	77.8	53.4	32.9	28.6	43.6	73.4	22.2	40.2	55.1	33.3	21.1	19.0	34.0	42.2	77.5	59.7	
BH7	1	5737.8	2490102.7	393.77	0	4.5	1.88	100.0	52.6	83.9	54.7	34.4	29.0	49.5	72.6	16.1	41.7	50.5	33.1	22.0	18.7	28.5	44.2	78.0	62.2	
	2				4.5	9.0	1.80	100.0	48.8	77.7	51.9	38.1	29.7	39.6	73.2	22.3	38.2	60.4	32.9	27.1	20.4	33.3	43.0	72.9	59.4	
	3				9	13.5	1.90	100.0	51.7	75.8	54.1	32.5	29.6	43.3	72.5	24.2	44.0	56.7	35.7	18.4	19.5	38.3	43.5	81.6	58.9	
	4				13.5	18.0	1.83	100.0	50.0	73.9	53.7	31.3	29.3	42.6	71.6	26.1	39.7	57.4	34.0	15.8	18.3	41.6	40.0	84.2	56.0	
BH10	1	5688.7	2490314.1	396.21	0	4.5	1.90	100.0	52.6	77.3	55.8	31.4	29.2	45.8	74.1	22.7	41.8	54.2	34.5	21.1	18.7	33.1	44.5	78.9	61.7	
	2				4.5	9.0	1.83	100.0	49.7	79.8	51.7	36.9	27.9	42.9	72.2	20.2	41.7	57.1	32.8	22.9	18.2	34.2	42.6	77.1	59.0	
	3				9	13.5	1.86	100.0	52.8	100.0	52.8	45.3	28.6	54.7	72.8					20.2	20.0	25.1	35.5	79.8	61.1	
	4				13.5	16.5	1.91	100.0	44.5	75.7	59.2	30.0	38.2	45.8	72.9	24.3	45.8	54.2	41.6	17.1	19.9	37.1	51.7	82.9	63.4	
BH11	1	5739.2	2490327.0	396.56	0	4.5	1.91	100.0	57.9	100.0	57.9	36.0	29.3	64.0	74.0					16.0	20.8	20.0	36.1	84.0	65.0	
	2				4.5	9.0	1.90	100.0	52.7	77.5	56.3	31.5	30.0	46.0	74.2	22.5	40.6	54.0	34.5	24.0	20.1	29.9	46.0	76.0	63.1	
	3				9	13.5	1.84	100.0	50.9	77.6	54.1	33.9	29.5	43.7	73.1	22.4	40.0	56.3	33.7	24.8	20.8	31.5	43.8	75.2	60.8	
	4				13.5	16.5	1.80	100.0	51.5	78.6	54.4	34.8	29.7	43.7	74.2	21.4	40.9	56.3	33.9	21.3	19.9	34.9	42.5	78.7	60.1	
BH12	1	5639.9	2490291.3	395.44	0	4.5	1.88	100.0	51.0	80.0	53.2	35.4	28.6	44.7	72.6	20.0	42.2	55.3	33.5	25.2	20.1	30.2	44.7	74.8	61.4	
	2				4.5	9.0	1.82	100.0	49.9	82.7	51.7	46.1	33.6	36.5	74.5	17.3	41.0	63.5	35.7	25.7	23.0	37.8	44.3	74.3	59.1	
	3				9	13.5	1.83	100.0	49.7	86.6	51.1	41.8	29.2	44.8	71.6	13.4	40.1	55.2	31.8	25.0	20.1	30.2	41.5	75.0	59.5	
	4				13.5	18.0	1.84	100.0	48.5	87.2	50.0	42.4	28.6	44.8	70.2	12.8	38.5	55.2	30.9	24.8	18.3	30.3	41.2	75.2	58.5	

Appendix 2: Discard Dump Borehole Results

Table A2.1 continued: Old Dump data – raw data

Borehole	Section	Co-ordinates			Depth (m)		Insitu Results				+0.5mm Material		+0.5mm Floats RD1.80		+0.5mm Sinks RD1.80		-0.5mm Material		Flotation Feed		Flotation Product		Flotation Tailings		Final Tailings		
		X	Y	Z	From	To	Sample RD	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	+0.5mm S1.80 + Flotation Tailings	Ash (%)
BH13	1	5629.9	2490254.9	395.18	0	4.5	1.81	100.0	48.3	72.5	52.2	33.7	29.0	38.8	72.3	27.5	38.3	61.2	33.2	32.7	19.0	28.6	49.3	67.3	62.6		
	2				4.5	9.0	1.86	100.0	49.0	74.0	52.3	33.9	27.6	40.1	73.2	26.0	39.6	59.9	32.8	27.7	17.7	32.2	45.9	72.3	61.0		
	3				9	13.5	1.81	100.0	51.8	96.3	52.3	42.8	28.3	53.5	71.4	3.7	38.6	46.5	29.1	20.2	18.2	26.3	37.5	79.8	60.2		
	4				13.5	18.0	1.85	100.0	51.0	75.2	54.1	31.4	29.2	43.8	72.0	24.8	41.5	56.2	34.6	24.5	20.4	31.7	45.6	75.5	60.9		
BH14	1	5633.9	2490207.5	395.92	0	4.5	1.81	100.0	48.4	73.3	52.2	33.0	29.1	40.4	71.0	26.7	37.9	59.6	33.1	29.1	17.6	30.5	47.9	70.9	61.1		
	2				4.5	9.0	1.81	100.0	48.9	74.4	52.1	35.3	28.8	39.1	73.1	25.6	39.7	60.9	33.4	28.8	18.7	32.2	46.5	71.2	61.1		
	3				9	13.5	1.78	100.0	42.3	57.8	44.0	38.8	29.7	19.0	73.2	42.2	40.0	81.0	35.0	37.1	18.4	43.9	49.1	62.9	56.4		
	4				13.5	16.0	1.89	100.0	53.6	71.7	56.4	28.5	29.8	43.2	73.9	28.3	46.4	56.8	38.1	22.0	19.7	34.7	49.7	78.0	63.1		
BH15	1	5637.3	2490151.9	395.29	0	4.5	1.82	100.0	48.7	75.7	52.4	35.4	29.7	40.2	72.4	24.3	37.1	59.8	32.7	26.9	18.1	32.8	44.7	73.1	60.0		
	2				4.5	9.0	1.80	100.0	48.0	77.0	50.9	37.3	28.3	39.7	72.1	23.0	38.3	60.3	32.1	29.6	18.5	30.7	45.2	70.4	60.4		
	3				9	13.5	1.85	100.0	49.7	72.2	53.1	31.8	28.7	40.5	72.3	27.8	40.9	59.5	34.4	29.9	19.1	29.7	49.8	70.1	62.8		
	4				13.5	15.0	1.89	100.0	51.6	77.8	53.7	33.0	30.3	44.8	70.9	22.2	44.3	55.2	35.9	23.6	21.4	31.6	46.7	76.4	60.9		
BH16	1	5636.8	2490100.4	393.6	0	4.5	1.83	100.0	48.2	73.4	51.5	34.6	29.1	38.8	71.6	26.6	38.9	61.2	33.4	28.2	18.4	33.0	46.2	71.8	59.9		
	2				4.5	9.0	1.83	100.0	48.2	72.3	51.9	33.8	28.3	38.4	72.8	27.7	38.4	61.6	32.8	33.1	17.9	28.5	50.2	66.9	63.2		
	3				9	13.5	1.76	100.0	49.4	72.7	53.9	33.5	29.2	39.2	75.0	27.3	37.4	60.8	32.9	35.2	17.8	25.7	53.6	64.8	66.6		
	4				13.5	18.0	1.79	100.0	48.2	73.8	51.6	34.4	28.3	39.5	71.8	26.2	38.7	60.5	32.8	30.4	18.0	30.1	47.8	69.6	61.4		
BH17	1	5690	2490086.3	392.7	0	4.5	1.80	100.0	51.0	74.7	54.4	32.5	29.2	42.2	73.8	25.3	41.1	57.8	34.4	25.6	17.0	32.2	48.3	74.4	62.7		
	2				4.5	9.0	1.79	100.0	49.4	76.2	53.0	35.8	29.6	40.4	73.6	23.8	37.9	59.6	32.9	31.8	17.4	27.8	50.6	68.2	64.3		
	3				9	13.5	1.80	100.0	50.4	76.1	53.8	35.0	30.3	41.1	73.8	23.9	39.7	58.9	34.1	23.6	19.0	35.3	44.2	76.4	60.1		
	4				13.5	18.0	1.86	100.0	52.1	75.6	55.1	32.0	30.3	43.6	73.4	24.4	42.6	56.4	35.6	17.8	19.0	38.6	43.3	82.2	59.2		
BH18	1	5688.5	2490053.4	390.38	0	4.5	1.87	100.0	50.7	75.4	54.5	32.3	29.4	43.0	73.4	24.6	38.8	57.0	33.4	26.8	18.1	30.2	47.1	73.2	62.6		
	2				4.5	9.0	1.83	100.0	49.0	77.3	51.9	37.0	28.9	40.3	73.0	22.7	39.0	59.7	32.7	28.5	18.4	31.2	45.8	71.5	61.2		
	3				9	13.5	1.86	100.0	49.7	70.1	53.4	31.2	30.3	39.0	71.9	29.9	40.9	61.0	35.5	28.0	19.7	33.0	48.9	72.0	61.4		
	4				13.5	15.0	1.86	100.0	53.4	73.1	57.9	26.9	29.9	46.2	74.1	26.9	41.4	53.8	35.7	25.2	20.4	28.6	49.1	74.8	64.5		
BH19	1	5640.8	2490054.1	390.02	0	4.5	1.82	100.0	49.7	77.1	52.9	35.7	29.1	41.4	73.4	22.9	39.0	58.6	33.0	27.8	18.0	30.8	46.5	72.2	61.9		
	2				4.5	9.0	1.82	100.0	49.0	73.9	53.2	34.3	29.8	39.6	73.6	26.1	36.8	60.4	32.8	33.5	19.1	26.9	50.0	66.5	64.0		
	3				9	13.5	1.80	100.0	49.5	67.4	54.0	30.0	29.3	37.4	73.8	32.6	40.3	62.6	35.0	30.4	19.0	32.1	50.1	69.6	62.9		
	4				13.5	15.0	1.82	100.0	48.9	78.3	51.8	38.5	29.5	39.8	73.4	21.7	38.6	60.2	32.7	27.7	19.2	32.5	44.2	72.3	60.3		
BH20	1	5604.6	2490065.4	390.87	0	4.5	1.87	100.0	50.2	78.7	53.1	36.9	30.4	41.8	73.2	21.3	39.6	58.2	33.8	29.0	19.8	29.2	47.7	71.0	62.7		
	2				4.5	9.0	1.80	100.0	48.4	68.5	53.1	32.4	30.3	36.1	73.5	31.5	38.2	63.9	34.2	21.9	20.3	42.0	41.4	78.1	56.2		
	3				9	13.5	1.80	100.0	50.1	75.2	53.4	32.8	29.6	42.4	71.9	24.8	40.1	57.6	34.1	25.8	20.2	31.8	45.4	74.2	60.5		
	4				13.5	15.0	1.75	100.0	46.7	67.7	51.6	32.5	28.3	35.2	73.1	32.3	36.3	64.8	32.3	27.3	16.6	37.5	43.7	72.7	57.9		
BH21	1	5588.5	2490099.4	393.03	0	4.5	1.72	100.0	47.7	73.9	51.9	36.8	29.3	37.1	74.4	26.1	35.6	62.9	31.9	30.6	17.9	32.3	45.2	69.4	60.8		
	2				4.5	9.0	1.75	100.0	48.4	72.6	51.7	35.2	28.8	37.4	73.1	27.4	40.0	62.6	33.7	27.5	18.3	35.1	45.7	72.5	59.9		
	3				9	13.5	1.75	100.0	48.6	70.3	52.4	32.0	29.9	38.4	71.1	29.7	39.5	61.6	34.5	35.5	20.2	26.1	54.0	64.5	64.2		
	4				13.5	16.5	1.85	100.0	50.0	80.2	52.0	38.0	28.6	42.2	73.0	19.8	42.0	57.8	33.2	28.4	21.1	29.3	44.8	71.6	61.5		
BH22	1	5590	2490149.4	395.12	0	4.5	1.83	100.0	49.5	73.6	53.2	35.5	29.6	38.1	75.3	26.4	26.4	61.9	33.6	35.4	21.5	26.4	49.9	64.6	64.9		
	2				4.5	9.0	1.83	100.0	48.4	76.9	50.8	36.5	28.3	40.4	71.2	23.1	23.1	59.6	33.0	28.8	18.1	30.8	46.9	71.2	60.7		
	3				9	12.0	1.84	100.0	48.6	71.1	52.9	32.4	29.0	38.6	73.0	28.9	28.9	61.4	33.1	34.6	19.8	26.8	50.4	65.4	63.8		
	4				0	4.5	1.77	100.0	48.3	72.7	51.5	35.7	29.3	37.0	72.9	27.3	27.3	63.0	33.8	27.2	18.9	35.8	45.1	72.8	59.3		
BH23	1	5591.7	2490200.4	395.71	0	4.5	1.83	100.0	47.7	74.7	51.1	36.0	28.9	38.7	71.6	25.3	25.3	61.3	32.6	25.2	19.4	36.1	41.7	74.8	57.2		
	2				4.5	9.0	1.83	100.0	48.0	72.2	51.7	34.9	28.3	37.3	73.6	27.8	27.8	62.7	32.8	22.7	18.2	39.9	41.1	77.3	56.8		
	3				9	13.5	1.83	100.0	47.6	76.7	50.4	37.7	28.4	39.0	71.7	23.3	23.3	61.0	32.2	27.9	18.2	33.1	43.9	72.1	58.9		
	4				13.5	15.0	1.78	100.0	47.6	76.7	50.4	37.7	28.4	39.0	71.7	23.3	23.3	61.0	32.2	27.9	18.2	33.1	43.9	72.1	58.9		
BH24	1	5591.3	2490251.0	394.56	0	4.5	1.83	100.0	51.6	76.7	53.6	36.2	29.8	40.5	74.8	23.3	23.3	59.5	35.7	25.3	19.9	34.2	47.4	74.7	62.3		
	2				4.5	9.0	1.76	100.0	49.8	72.7	53.7	33.0	30.1	39.7	73.3	27.3	27.3	60.3	34.3	25.0	19.0	35.3	45.1	75.0	60.0		
	3				9	13.5	1.83	100.0	53.1	64.9	57.2	24.9	30.0	40.0	74.1	35.1	35.1	60.0	39.0	24.7	20.3	35.3	52.1	75.3	63.8		
	4				13.5	16.5	1.96	100.0	57.5	72.0	61.1	23.5	30.5	48.5	76.0	28.0	28.0	51.5	40.1	15.1	20.5	36.5	48.1	84.9	64.0		
Summary Statistics	Average	-	-	-	-	-	1.84	100.0	50.3	76.9	53.5	34.5	29.8	42.3	72.9	23.6	38.6	57.9	34.1	25.3	19.3	31.8	45.9	74.1	61		

Appendix 2: Discard Dump Borehole Results

Table A2.2: Old Dump data – Adjusted data

Borehole	Section	Co-ordinates			Depth (m)		Insitu Results				+0.5mm Material		+0.5mm Floats RD1.80		+0.5mm Sinks RD1.80		-0.5mm Material		Flotation Feed		Flotation Product		Flotation Tailings		Final Tailings	
		X	Y	Z	From	To	Sample RD	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	(+0.5mm F1.80) + (-0.5mm)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	+0.5mm S1.80 + Flotation Tailings	Yield (%)
BH1	1	5789.4	2490349.1	395.59	0	4.5	1.94	100.0	61.6	80.0	65.0	18.5	38.6	61.5	73.0	20.0	48.1	35.5	43.5	12.6	21.0	25.0	54.4	87.4	67.5	
	2				4.5	9.0	1.81	100.0	53.4	77.9	57.1	31.3	33.3	46.6	73.1	22.1	40.5	53.4	36.3	11.0	19.7	42.4	40.6	89.0	57.6	
	3				9	13.5	1.87	100.0	52.5	82.5	55.1	35.5	32.4	47.0	72.3	17.5	40.1	53.0	34.9	19.2	19.5	16.3	47.6	63.3	66.0	
	4				13.5	18.0	1.90	100.0	53.7	79.2	57.3	31.3	32.9	47.9	73.4	20.8	39.7	52.1	35.6	21.6	19.6	30.5	46.9	78.4	63.1	
BH2	1	5791.7	2490300.4	394.81	0	4.5	1.89	100.0	51.4	79.1	53.9	32.8	29.0	46.3	71.5	20.9	42.0	53.7	34.1	28.1	21.4	25.5	48.0	71.9	63.2	
	2				4.5	9.0	1.94	100.0	54.1	80.7	56.3	27.7	28.8	53.0	70.6	19.3	45.1	47.0	35.5	38.3	29.9	8.7	60.2	61.7	69.2	
	3				9	13.5	1.87	100.0	50.0	78.6	52.6	34.1	28.8	44.5	70.9	21.4	40.1	55.5	33.2	25.5	20.9	30.0	43.7	74.5	59.9	
	4				13.5	17.5	1.89	100.0	52.8	79.1	54.9	30.5	28.5	48.7	71.4	20.9	44.9	51.3	35.1	20.3	20.0	31.1	45.0	79.7	61.1	
BH3	1	5787.5	2490251.9	394.46	0	4.5	1.88	100.0	54.5	76.9	58.2	30.1	34.9	46.8	73.1	23.1	42.5	53.2	38.2	20.8	18.4	32.4	50.9	79.2	64.0	
	2				4.5	9.0	1.84	100.0	51.5	77.7	55.2	34.4	32.4	43.3	73.3	22.3	38.4	56.7	34.8	20.1	17.2	36.6	44.5	79.9	60.1	
	3				9	13.5	1.80	100.0	50.3	80.8	52.7	38.4	31.6	42.4	71.9	19.2	40.3	57.6	34.5	22.8	17.7	34.8	45.4	77.2	60.0	
	4				13.5	18.0	1.86	100.0	52.5	81.4	55.1	35.8	33.5	45.6	72.1	18.6	41.0	54.4	36.1	22.4	18.7	32.0	48.2	77.6	62.2	
BH4	1	5787.5	2490201.6	394.63	0	4.5	1.83	100.0	50.4	75.7	53.8	33.4	29.3	42.4	73.1	24.3	39.7	57.6	33.7	25.0	18.2	32.6	45.6	75.0	61.2	
	2				4.5	9.0	1.86	100.0	50.6	80.5	53.2	36.0	29.4	44.5	72.4	19.5	40.0	55.5	33.1	24.2	19.6	31.3	43.6	75.8	60.5	
	3				9	13.5	1.85	100.0	49.1	82.3	50.8	39.7	28.7	42.6	71.5	17.7	40.8	57.4	32.4	21.4	18.2	36.0	40.8	78.6	57.4	
	4				13.5	18.0	1.84	100.0	49.5	75.8	53.4	33.5	29.4	42.3	72.3	24.2	37.5	57.7	32.8	24.1	18.3	33.6	43.3	75.9	59.5	
BH5	1	5791.7	2490151.8	394.63	0	4.5	1.85	100.0	51.2	74.2	54.9	31.0	29.3	43.3	73.3	25.8	40.6	56.7	34.4	23.9	18.7	32.8	45.9	76.1	61.5	
	2				4.5	9.0	1.84	100.0	50.0	79.9	52.6	36.3	29.0	43.6	72.2	20.1	39.9	56.4	32.8	24.7	19.5	31.7	43.2	75.3	60.0	
	3				9	13.5	1.81	100.0	48.8	78.4	51.8	39.8	30.7	38.6	73.6	21.6	37.6	61.4	33.1	31.6	20.2	29.8	46.8	68.4	61.9	
	4				13.5	18.0	1.81	100.0	48.7	76.2	52.3	39.5	32.2	36.7	74.0	23.8	37.0	63.3	34.0	31.1	20.5	32.1	47.1	68.9	61.5	
BH6 (BH17a)	1	5681.6	2490075.9	392.19	0	4.5	1.81	100.0	52.9	80.7	55.6	33.5	29.3	42.2	73.8	19.3	42.0	52.8	33.9	20.5	18.0	32.3	44.1	74.5	60.9	
	2				4.5	9.0	1.90	100.0	51.0	82.8	53.0	35.8	28.9	40.4	73.6	17.2	41.1	53.0	32.9	21.8	18.0	31.3	43.2	71.7	60.3	
	3				9	13.5	1.88	100.0	49.2	79.6	51.6	37.0	28.1	41.1	73.8	20.4	39.9	57.4	32.3	22.9	17.2	34.5	42.3	75.6	59.5	
	4				13.5	18.0	1.88	100.0	50.5	77.8	53.4	32.9	28.6	43.6	73.4	22.2	40.2	55.1	33.3	21.1	19.0	34.0	42.2	77.5	59.7	
BH7	1	5737.8	2490102.7	393.77	0	4.5	1.88	100.0	52.6	83.9	54.7	34.4	29.0	49.5	72.6	16.1	41.7	50.5	33.1	22.0	18.7	28.5	44.2	78.0	62.2	
	2				4.5	9.0	1.80	100.0	48.8	77.7	51.9	38.1	29.7	39.6	73.2	22.3	38.2	60.4	32.9	27.1	20.4	33.3	43.0	72.9	59.4	
	3				9	13.5	1.90	100.0	51.7	75.8	54.1	32.5	29.6	43.3	72.5	24.2	44.0	56.7	35.7	18.4	19.5	38.3	43.5	81.6	58.9	
	4				13.5	18.0	1.83	100.0	50.0	73.9	53.7	31.3	29.3	42.6	71.6	26.1	39.7	57.4	34.0	15.8	18.3	41.6	40.0	84.2	56.0	
BH10	1	5688.7	2490314.1	396.21	0	4.5	1.90	100.0	52.6	77.3	55.8	31.4	29.2	45.8	74.1	22.7	41.8	54.2	34.5	21.1	18.7	33.1	44.5	78.9	61.7	
	2				4.5	9.0	1.83	100.0	49.7	79.8	51.7	36.9	27.9	42.9	72.2	20.2	41.7	57.1	32.8	22.9	18.2	34.2	42.6	77.1	59.0	
	3				9	13.5	1.86	100.0	52.8																	
	4				13.5	16.5	1.91	100.0	44.5	75.7	59.2	30.0	38.2	45.8	72.9	24.3	45.8	54.2	41.6	17.1	19.9	37.1	51.7	82.9	63.4	
BH11	1	5739.2	2490327.0	396.56	0	4.5	1.91	100.0	57.9																	
	2				4.5	9.0	1.90	100.0	52.7	77.5	56.3	31.5	30.0	46.0	74.2	22.5	40.6	54.0	34.5	24.0	20.1	29.9	46.0	76.0	63.1	
	3				9	13.5	1.84	100.0	50.9	77.6	54.1	33.9	29.5	43.7	73.1	22.4	40.0	56.3	33.7	24.8	20.8	31.5	43.8	75.2	60.8	
	4				13.5	16.5	1.80	100.0	51.5	78.6	54.4	34.8	29.7	43.7	74.2	21.4	40.9	56.3	33.9	21.3	19.9	34.9	42.5	78.7	60.1	
BH12	1	5639.9	2490291.3	395.44	0	4.5	1.88	100.0	51.0	80.0	53.2	35.4	28.6	44.7	72.6	20.0	42.2	55.3	33.5	25.2	20.1	30.2	44.7	74.8	61.4	
	2				4.5	9.0	1.82	100.0	49.9	82.7	51.7	46.1	33.6	36.5	74.5	17.3	41.0	63.5	35.7	25.7	23.0	37.8	44.3	74.3	59.1	
	3				9	13.5	1.83	100.0	49.7	86.6	51.1	41.8	29.2	44.8	71.6	13.4	40.1	55.2	31.8	25.0	20.1	30.2	41.5	75.0	59.5	
	4				13.5	18.0	1.84	100.0	48.5	87.2	50.0	42.4	28.6	44.8	70.2	12.8	38.5	55.2	30.9	24.8	18.3	30.3	41.2	75.2	58.5	

Appendix 2: Discard Dump Borehole Results

Table A2.2: Old Dump data – Adjusted data continued

Borehole	Section	Co-ordinates			Depth (m)		Insitu Results				+0.5mm Material		+0.5mm Floats RD1.80		+0.5mm Sinks RD1.80		-0.5mm Material		Flotation Feed		Flotation Product		Flotation Tailings		Final Tailings			
		X	Y	Z	From	To	Sample RD	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	+0.5mm S1.80 + Flotation Tailings	Yield (%)	Ash (%)		
BH13	1	5629.9	2490254.9	395.18	0	4.5	1.81	100.0	48.3	72.5	52.2	33.7	29.0	38.8	72.3	27.5	38.3	61.2	33.2	32.7	19.0	28.6	49.3	67.3	62.6			
	2				4.5	9.0	1.86	100.0	49.0	74.0	52.3	33.9	27.6	40.1	73.2	26.0	39.6	59.9	32.8	27.7	17.7	32.2	45.9	72.3	61.0			
	3				9	13.5	1.81	100.0	51.8	96.3	52.3	42.8	28.3	53.5	71.4	3.7	38.6	46.5	29.1	20.2	18.2	26.3	37.5	79.8	60.2			
	4				13.5	18.0	1.85	100.0	51.0	75.2	54.1	31.4	29.2	43.8	72.0	24.8	41.5	56.2	34.6	24.5	20.4	31.7	45.6	75.5	60.9			
BH14	1	5633.9	2490207.5	395.92	0	4.5	1.81	100.0	48.4	73.3	52.2	33.0	29.1	40.4	71.0	26.7	37.9	59.6	33.1	29.1	17.6	30.5	47.9	70.9	61.1			
	2				4.5	9.0	1.81	100.0	48.9	74.4	52.1	35.3	28.8	39.1	73.1	25.6	39.7	60.9	33.4	28.8	18.7	32.2	46.5	71.2	61.1			
	3				9	13.5	1.78	100.0	42.3	57.8	44.0	38.8	29.7	19.0	73.2	42.2	40.0	81.0	35.0	37.1	18.4	43.9	49.1	62.9	56.4			
	4				13.5	16.0	1.89	100.0	53.6	71.7	56.4	28.5	29.8	43.2	73.9	28.3	46.4	56.8	38.1	22.0	19.7	34.7	49.7	78.0	63.1			
BH15	1	5637.3	2490151.9	395.29	0	4.5	1.82	100.0	48.7	75.7	52.4	35.4	29.7	40.2	72.4	24.3	37.1	59.8	32.7	26.9	18.1	32.8	44.7	73.1	60.0			
	2				4.5	9.0	1.80	100.0	48.0	77.0	50.9	37.3	28.3	39.7	72.1	23.0	38.3	60.3	32.1	29.6	18.5	30.7	45.2	70.4	60.4			
	3				9	13.5	1.85	100.0	49.7	72.2	53.1	31.8	28.7	40.5	72.3	27.8	40.9	59.5	34.4	29.9	19.1	29.7	49.8	70.1	62.8			
	4				13.5	15.0	1.89	100.0	51.6	77.8	53.7	33.0	30.3	44.8	70.9	22.2	44.3	55.2	35.9	23.6	21.4	31.6	46.7	76.4	60.9			
BH16	1	5636.8	2490100.4	393.6	0	4.5	1.83	100.0	48.2	73.4	51.5	34.6	29.1	38.8	71.6	26.6	38.9	61.2	33.4	28.2	18.4	33.0	46.2	71.8	59.9			
	2				4.5	9.0	1.83	100.0	48.2	72.3	51.9	33.8	28.3	38.4	72.8	27.7	38.4	61.6	32.8	33.1	17.9	28.5	50.2	66.9	63.2			
	3				9	13.5	1.76	100.0	49.4	72.7	53.9	33.5	29.2	39.2	75.0	27.3	37.4	60.8	32.9	35.2	17.8	25.7	53.6	64.8	66.6			
	4				13.5	18.0	1.79	100.0	48.2	73.8	51.6	34.4	28.3	39.5	71.8	26.2	38.7	60.5	32.8	30.4	18.0	30.1	47.8	69.6	61.4			
BH17	1	5690	2490086.3	392.7	0	4.5	1.80	100.0	51.0	74.7	54.4	32.5	29.2	42.2	73.8	25.3	41.1	57.8	34.4	25.6	17.0	32.2	48.3	74.4	62.7			
	2				4.5	9.0	1.79	100.0	49.4	76.2	53.0	35.8	29.6	40.4	73.6	23.8	37.9	59.6	32.9	31.8	17.4	27.8	50.6	68.2	64.3			
	3				9	13.5	1.80	100.0	50.4	76.1	53.8	35.0	30.3	41.1	73.8	23.9	39.7	58.9	34.1	23.6	19.0	35.3	44.2	76.4	60.1			
	4				13.5	18.0	1.86	100.0	52.1	75.6	55.1	32.0	30.3	43.6	73.4	24.4	42.6	56.4	35.6	17.8	19.0	38.6	43.3	82.2	59.2			
BH18	1	5688.5	2490053.4	390.38	0	4.5	1.87	100.0	50.7	75.4	54.5	32.3	29.4	43.0	73.4	24.6	38.8	57.0	33.4	26.8	18.1	30.2	47.1	73.2	62.6			
	2				4.5	9.0	1.83	100.0	49.0	77.3	51.9	37.0	28.9	40.3	73.0	22.7	39.0	59.7	32.7	28.5	18.4	31.2	45.8	71.5	61.2			
	3				9	13.5	1.86	100.0	49.7	70.1	53.4	31.2	30.3	39.0	71.9	29.9	40.9	61.0	35.5	28.0	19.7	33.0	48.9	72.0	61.4			
	4				13.5	15.0	1.86	100.0	53.4	73.1	57.9	26.9	29.9	46.2	74.1	26.9	41.4	53.8	35.7	25.2	20.4	28.6	49.1	74.8	64.5			
BH19	1	5640.8	2490054.1	390.02	0	4.5	1.82	100.0	49.7	77.1	52.9	35.7	29.1	41.4	73.4	22.9	39.0	58.6	33.0	27.8	18.0	30.8	46.5	72.2	61.9			
	2				4.5	9.0	1.82	100.0	49.0	73.9	53.2	34.3	29.8	39.6	73.6	26.1	36.8	60.4	32.8	33.5	19.1	26.9	50.0	66.5	64.0			
	3				9	13.5	1.80	100.0	49.5	67.4	54.0	30.0	29.3	37.4	73.8	32.6	40.3	62.6	35.0	30.4	19.0	32.1	50.1	69.6	62.9			
	4				13.5	15.0	1.82	100.0	48.9	78.3	51.8	38.5	29.5	39.8	73.4	21.7	38.6	60.2	32.7	27.7	19.2	32.5	44.2	72.3	60.3			
BH20	1	5604.6	2490065.4	390.87	0	4.5	1.87	100.0	50.2	78.7	53.1	36.9	30.4	41.8	73.2	21.3	39.6	58.2	33.8	29.0	19.8	29.2	47.7	71.0	62.7			
	2				4.5	9.0	1.80	100.0	48.4	68.5	53.1	32.4	30.3	36.1	73.5	31.5	38.2	63.9	34.2	21.9	20.3	42.0	41.4	78.1	56.2			
	3				9	13.5	1.80	100.0	50.1	75.2	53.4	32.8	29.6	42.4	71.9	24.8	40.1	57.6	34.1	25.8	20.2	31.8	45.4	74.2	60.5			
	4				13.5	15.0	1.75	100.0	46.7	67.7	51.6	32.5	28.3	35.2	73.1	32.3	36.3	64.8	32.3	27.3	16.6	37.5	43.7	72.7	57.9			
BH21	1	5588.5	2490099.4	393.03	0	4.5	1.72	100.0	47.7	73.9	51.9	36.8	29.3	37.1	74.4	26.1	35.6	62.9	31.9	30.6	17.9	32.3	45.2	69.4	60.8			
	2				4.5	9.0	1.75	100.0	48.4	72.6	51.7	35.2	28.8	37.4	73.1	27.4	40.0	62.6	33.7	27.5	18.3	35.1	45.7	72.5	59.9			
	3				9	13.5	1.75	100.0	48.6	70.3	52.4	32.0	29.9	38.4	71.1	29.7	39.5	61.6	34.5	35.5	20.2	26.1	54.0	64.5	64.2			
	4				13.5	16.5	1.85	100.0	50.0	80.2	52.0	38.0	28.6	42.2	73.0	19.8	42.0	57.8	33.2	28.4	21.1	29.3	44.8	71.6	61.5			
BH22	1	5590	2490149.4	395.12	0	4.5	1.83	100.0	49.5	73.6	53.2	35.5	29.6	38.1	75.3	26.4	26.4	61.9	33.6	35.4	21.5	26.4	49.9	64.6	64.9			
	2				4.5	9.0	1.83	100.0	48.4	76.9	50.8	36.5	28.3	40.4	71.2	23.1	23.1	59.6	33.0	28.8	18.1	30.8	46.9	71.2	60.7			
	3				9	12.0	1.84	100.0	48.6	71.1	52.9	32.4	29.0	38.6	73.0	28.9	28.9	61.4	33.1	34.6	19.8	26.8	50.4	65.4	63.8			
	4				0	4.5	1.77	100.0	48.3	72.7	51.5	35.7	29.3	37.0	72.9	27.3	27.3	63.0	33.8	27.2	18.9	35.8	45.1	72.8	59.3			
BH23	1	5591.7	2490200.4	395.71	0	4.5	1.83	100.0	47.7	74.7	51.1	36.0	28.9	38.7	71.6	25.3	25.3	61.3	32.6	25.2	19.4	36.1	41.7	74.8	57.2			
	2				9	13.5	1.83	100.0	48.0	72.2	51.7	34.9	28.3	37.3	73.6	27.8	27.8	62.7	32.8	22.7	18.2	39.9	41.1	77.3	56.8			
	3				13.5	15.0	1.78	100.0	47.6	76.7	50.4	37.7	28.4	39.0	71.7	23.3	23.3	61.0	32.2	27.9	18.2	33.1	43.9	72.1	58.9			
	4				0	4.5	1.83	100.0	51.6	76.7	53.6	36.2	29.8	40.5	74.8	23.3	23.3	59.5	35.7	25.3	19.9	34.2	47.4	74.7	62.3			
BH24	1	5591.3	2490251.0	394.56	0	4.5	1.80	100.0	49.8	72.7	53.7	33.0	30.1	39.7	73.3	27.3	27.3	60.3	34.3	25.0	19.0	35.3	45.1	75.0	60.0			
	2				9	13.5	1.83	100.0	53.1	64.9	57.2	24.9	30.0	40.0	74.1	35.1	35.1	60.0	39.0	24.7	20.3	35.3	52.1	75.3	63.8			
	3				13.5	16.5	1.96	100.0	57.5	72.0	61.1	23.5	30.5	48.5	76.0	28.0	28.0	51.5	40.1	15.1	20.5	36.5	48.1	84.9	64.0			
	4				0	4.5	1.84	100.0	50.3	76.4	53.4	34.3	29.8	41.9	72.9	23.6	38.6	57.9	34.1	25.5	19.3	32.3	45.9	74.2	61.1			
Summary Statistics	Average	-	-	-	-	-	-	-	-	1.84	100.0	50.3	76.4	53.4	34.3	29.8	41.9	72.9	23.6	38.6	57.9	34.1	25.5	19.3	32.3	45.9	74.2	61.1
	Std Dev	-	-	-	-	-	-	-	-	0.05	0.0	2.8	5.0	3.0	4.4	1.9	5.5	1.1	5.0	5.1	5.6	2.1	5.2	1.8	5.4	3.8	5.3	2.9
	Max	-	-	-	-	-	-	-	-	1.96	100.																	

Table A2.3: Old Dump raw ash data – descriptive statistics all data

Description	Value
Mean	50.3
Standard Error	0.3
Median	50.0
Standard Deviation	2.8
Sample Variance	8.0
Kurtosis	4.3
Skewness	0.4
Range	21.3
Minimum	40.4
Maximum	61.6
Sum	4529.5
Count	90
Largest(1)	61.6
Smallest(1)	40.4

Table A2.4: Old Dump raw ash data – descriptive statistics by layer depth

Description	Layer 1	Layer 2	Layer 3	Layer 4	Layer 5
Mean	51.3	49.8	49.8	50.6	48.8
Standard Error	0.7	0.4	0.5	0.6	4.2
Median	50.8	49.2	49.7	50.5	53.0
Standard Deviation	3.3	1.8	2.2	2.9	7.3
Sample Variance	11.0	3.1	4.8	8.3	53.3
Kurtosis	3.8	0.8	6.1	0.7	-
Skewness	1.8	1.2	-1.7	0.2	-1.7
Range	14.0	6.4	10.8	13.0	12.7
Minimum	47.7	47.7	42.3	44.5	40.4
Maximum	61.6	54.1	53.1	57.5	53.0
Sum	1128.5	1095.9	1096.0	1062.7	146.4
Count	22.0	22.0	22.0	21.0	3.0
Largest(1)	61.6	54.1	53.1	57.5	53.0
Smallest(1)	47.7	47.7	42.3	44.5	40.4

A2.2 Old Dump Data

The results for the boreholes drilling on the accessible section of the new discard dump are presented in Table A2.5. overleaf. The descriptive statistics are presented globally in Table A2.6 and by layer in Table A2.7 respectively.

Appendix 2: Discard Dump Borehole Results

Table A2.5: Old Dump data – raw data

Borehole	Section	Depth (m)		Insitu Results			+0.5mm Material		+0.5mm Floats RD1.80		+0.5mm Sinks RD1.80		-0.5mm Material		Flotation Feed		Flotation Product		Flotation Tailings		Final Tailings	
		From	To	Sample RD	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	(+0.5mm F1.80) + (-0.5mm)		Yield (%)	Ash (%)	Yield (%)	Ash (%)	+0.5mm S1.80 + Flotation Tailings	
															Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
BH25	1	0	4.5	1.89	100.0	56.9	82.3	59.5	26.3	28.8	56.0	73.9	17.7	44.8	44.0	35.3	21.5	20.9	22.5	48.9	78.5	66.8
	2	4.5	9.0	1.86	100.0	52.4	81.0	55.3	33.0	29.1	48.0	73.3	19.0	40.2	52.1	33.2	13.1	20.5	39.0	37.4	86.9	57.2
	3	9	13.5	1.90	100.0	57.8	82.8	60.1	25.0	29.8	57.8	73.1	17.2	47.0	42.2	36.8	18.6	22.2	23.6	48.4	81.4	66.0
BH26	1	0	4.5	1.89	100.0	56.4	80.9	59.7	26.9	31.2	54.0	73.9	19.1	42.4	46.0	35.8	20.7	19.3	25.3	49.4	79.3	66.1
	2	4.5	9.0	1.89	100.0	56.0	80.8	59.4	27.1	31.0	53.7	73.8	19.2	41.8	46.3	35.5	22.7	22.0	23.6	48.5	77.3	66.0
	3	9	13.5	1.88	100.0	55.4	82.8	58.6	30.6	31.0	52.2	74.7	17.2	40.3	47.8	34.3	21.4	20.9	26.4	45.2	78.6	64.8
BH27	1	0	4.5	1.87	100.0	54.4	81.3	57.1	29.4	30.3	51.9	72.2	18.7	42.8	48.1	35.1	18.6	19.4	29.5	45.1	81.4	62.4
	2	4.5	9.0	1.88	100.0	54.6	79.4	57.7	27.6	30.3	51.8	72.3	20.6	42.7	48.2	35.6	16.8	19.0	31.4	44.5	83.2	61.8
BH28	1	0	4.5	1.90	100.0	58.0	83.0	60.8	25.6	31.5	57.4	73.8	17.0	44.6	42.6	36.7	18.6	21.9	24.0	48.2	81.4	66.3
	2	4.5	9.0	1.89	100.0	56.0	83.0	58.7	28.7	31.0	54.3	73.3	17.0	43.0	45.7	35.5	15.9	17.3	29.8	45.1	84.1	63.3
Summary Statistics	Average	-	-	1.89	100.0	55.8	81.7	58.7	28.0	30.4	53.7	73.4	18.3	42.9	46.3	35.4	18.8	20.3	27.5	46.1	81.2	64.1
	Std Dev	-	-	0.01	0.0	1.7	1.2	1.6	2.5	0.9	2.9	0.8	1.2	2.1	3.0	1.1	3.0	1.6	5.1	3.6	3.0	3.0
	Max	-	-	1.90	100.0	58.0	83.0	60.8	33.0	31.5	57.8	74.7	20.6	47.0	52.1	36.8	22.7	22.2	39.0	49.4	86.9	66.8
	Min	-	-	1.86	100.0	52.4	79.4	55.3	25.0	28.8	48.0	72.2	17.0	40.2	42.2	33.2	13.1	17.3	22.5	37.4	77.3	57.2

Table A2.6: New Dump raw ash data – descriptive statistics all data

Description	Value
Mean	55.8
Standard Error	0.5
Median	56.0
Standard Deviation	1.7
Sample Variance	2.8
Kurtosis	0.5
Skewness	-0.7
Range	5.6
Minimum	52.4
Maximum	58.0
Sum	557.9
Count	10
Largest(1)	58.0
Smallest(1)	52.4

Table A2.7: New Dump raw ash data – descriptive statistics by layer depth

Description	Layer 1	Layer 2	Layer 3
Mean	56.4	54.8	56.6
Standard Error	0.8	0.9	1.2
Median	56.7	55.3	56.6
Standard Deviation	1.5	1.7	1.7
Sample Variance	2.3	2.9	2.9
Kurtosis	1.4	0.6	
Skewness	-0.8	-1.2	
Range	3.6	3.6	2.4
Minimum	54.4	52.4	55.4
Maximum	58.0	56.0	57.8
Sum	225.7	219.0	113.2
Count	4	4	2
Largest(1)	58.0	56.0	57.8
Smallest(1)	54.4	52.4	55.4

A2.3 Flotation Rate Data on All Samples

This section contained the flotation rate test results from all of the tests performed on the old dump (including the fines section in the centre), the new (operational) dump and the “Visgat” dam which contained fine material.

Appendix 2: Discard Dump Borehole Results

Table A2.8: Flotation rate test results

Sample	Stream	Time (min)	Sample 1				Sample 2				Sample 3				Sample 4				Sample 5			
			Fractional		Cumulative		Fractional		Cumulative		Fractional		Cumulative		Fractional		Cumulative		Fractional		Cumulative	
			Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)
BH1	Conc 1	0.50	15.2	20.8	15.2	20.8	15.5	19.3	15.5	19.3	16.1	19.7	16.1	19.7	20.6	19.4	20.6	19.4				
	Conc 2	1.00	6.4	20.6	21.6	20.7	5.5	19.1	21.0	19.2	6.9	19.0	23.0	19.5	6.6	18.7	27.2	19.2				
	Conc 3	2.00	6.2	21.2	27.8	20.8	7.8	19.3	28.8	19.3	7.0	19.1	30.1	19.4	8.3	19.5	35.5	19.3				
	Conc 4	4.00	4.8	22.1	32.6	21.0	6.4	21.5	35.2	19.7	6.2	19.7	36.2	19.4	6.0	21.2	41.4	19.6				
	Tails	4.00	67.4	54.4	-	-	64.8	40.6	-	-	63.8	47.6	-	-	58.6	46.9	-	-				
	Feed		100.0	43.5	100.0	43.5	100.0	36.3	100.0	36.3	100.0	34.9	100.0	34.9	100.0	35.6	100.0	35.6				
BH2	Conc 1	0.50	24.9	21.8	24.9	21.8	52.8	29.8	52.8	29.8	15.2	17.8	15.2	17.8	19.2	19.8	19.2	19.8	17.8	25.9	17.8	25.9
	Conc 2	1.00	11.0	20.8	35.9	21.5	21.0	30.3	73.8	29.9	5.6	19.0	20.8	18.1	7.0	19.5	26.2	19.7	5.7	26.4	23.5	26.0
	Conc 3	2.00	10.7	20.6	46.6	21.3	9.2	31.0	83.0	30.0	12.0	22.8	32.9	19.8	7.5	20.0	33.8	19.8	6.0	27.1	29.5	26.2
	Conc 4	4.00	5.8	22.7	52.4	21.4	5.9	39.1	88.9	30.6	8.1	23.9	40.9	20.6	5.7	21.3	39.5	20.0	4.1	28.6	33.6	26.5
	Tails	4.00	47.6	48.0	-	-	11.1	60.2	-	-	59.1	43.7	-	-	60.5	45.0	-	-	66.4	35.9	-	-
	Feed		100.0	34.1	100.0	34.1	100.0	35.5	100.0	35.5	100.0	34.3	100.0	34.3	100.0	35.1	100.0	35.1	100.0	32.8	100.0	32.8
BH3	Conc 1	0.50	19.5	19.6	19.5	19.6	16.6	17.4	16.6	17.4	19.3	17.9	19.3	17.9	21.6	18.4	21.6	18.4	17.0	17.8	17.0	17.8
	Conc 2	1.00	7.7	16.8	27.2	18.8	6.0	16.7	22.6	17.2	8.3	17.0	27.6	17.6	6.4	18.0	27.9	18.3	6.7	17.5	23.7	17.7
	Conc 3	2.00	6.4	16.9	33.6	18.4	6.5	16.7	29.1	17.1	7.1	17.3	34.7	17.6	7.8	18.9	35.7	18.4	7.0	18.1	30.7	17.8
	Conc 4	4.00	5.5	18.3	39.1	18.4	6.3	17.6	35.4	17.2	4.9	18.8	39.6	17.7	5.4	20.6	41.1	18.7	4.8	19.3	35.4	18.0
	Tails	4.00	60.9	50.9	-	-	64.6	44.5	-	-	60.4	45.4	-	-	58.9	48.2	-	-	64.6	41.2	-	-
	Feed		100.0	38.2	100.0	38.2	100.0	34.8	100.0	34.8	100.0	34.5	100.0	34.5	100.0	36.1	100.0	36.1	100.0	33.0	100.0	33.0
BH4	Conc 1	0.50	21.6	18.2	21.6	18.2	21.1	19.4	21.1	19.4	18.5	17.8	18.5	17.8	14.1	17.8	14.1	17.8	22.1	18.4	22.1	18.4
	Conc 2	1.00	5.6	17.0	27.2	18.0	8.2	18.7	29.3	19.2	6.0	17.3	24.5	17.7	8.1	17.4	22.2	17.7	8.2	17.9	30.3	18.3
	Conc 3	2.00	9.4	17.8	36.6	17.9	8.3	19.7	37.5	19.3	6.9	18.5	31.4	17.9	11.7	18.2	33.9	17.8	8.5	18.0	38.9	18.2
	Conc 4	4.00	6.8	19.6	43.4	18.2	6.0	21.5	43.5	19.6	5.8	20.1	37.2	18.2	7.9	20.0	41.8	18.3	5.5	20.1	44.3	18.4
	Tails	4.00	56.6	45.6	-	-	56.5	43.6	-	-	62.8	40.8	-	-	58.2	43.3	-	-	55.7	45.7	-	-
	Feed		100.0	33.7	100.0	33.7	100.0	32.6	100.0	32.6	100.0	32.4	100.0	32.4	100.0	32.8	100.0	32.8	100.0	33.6	100.0	33.6
BH5	Conc 1	0.50	21.6	18.6	21.6	18.6	21.1	19.8	21.1	19.8	28.0	19.6	28.0	19.6	18.0	20.1	18.0	20.1				
	Conc 2	1.00	6.7	18.0	28.3	18.5	8.9	18.6	30.0	19.4	9.1	19.6	37.1	19.6	11.9	20.0	29.9	20.1				
	Conc 3	2.00	7.7	18.6	36.0	18.5	7.7	18.9	37.7	19.3	8.3	21.0	45.4	19.9	11.2	20.2	41.2	20.1				
	Conc 4	4.00	6.2	20.2	42.2	18.7	6.0	20.5	43.7	19.5	6.0	23.1	51.4	20.2	8.0	22.3	49.2	20.5				
	Tails	4.00	57.8	45.9	-	-	56.3	43.2	-	-	48.6	46.8	-	-	50.8	47.1	-	-				
	Feed		100.0	34.4	100.0	34.4	100.0	32.8	100.0	32.8	100.0	33.1	100.0	33.1	100.0	34.0	100.0	34.0				
BH6 (BH17a)	Conc 1	0.50	15.3	17.8	15.3	17.8	19.6	17.9	19.6	17.9	19.4	16.9	19.4	16.9	16.2	19.1	16.2	19.1				
	Conc 2	1.00	8.9	17.1	24.2	17.5	8.0	17.4	27.6	17.8	5.0	16.5	24.4	16.8	7.1	18.0	23.2	18.8				
	Conc 3	2.00	7.7	17.9	31.9	17.6	7.2	17.9	34.9	17.8	7.5	17.0	32.0	16.9	8.2	18.6	31.4	18.7				
	Conc 4	4.00	7.0	19.4	38.9	18.2	6.2	19.4	41.0	18.0	7.9	18.4	39.9	17.2	6.9	20.0	38.4	19.0				
	Tails	4.00	61.1	44.1	-	-	59.0	43.2	-	-	60.1	42.3	-	-	61.6	47.1	-	-				
	Feed		100.0	33.9	100.0	33.9	100.0	32.9	100.0	32.9	100.0	32.3	100.0	32.3	100.0	36.3	100.0	36.3				
BH7	Conc 1	0.50	21.0	18.3	21.0	18.3	20.7	20.4	20.7	20.4	13.2	19.3	13.2	19.3	11.4	16.6	11.4	16.6				
	Conc 2	1.00	7.9	18.1	28.9	18.2	8.8	20.0	29.6	20.3	6.4	19.0	19.6	19.2	6.2	16.3	17.6	16.5				
	Conc 3	2.00	9.2	19.2	38.1	18.5	8.4	20.4	38.0	20.3	6.5	19.4	26.2	19.3	12.4	17.7	30.0	17.0				
	Conc 4	4.00	5.6	20.5	43.6	18.7	6.9	21.2	44.9	20.4	6.3	20.5	32.4	19.5	8.8	22.6	38.8	18.3				
	Tails	4.00	56.4	44.2	-	-	55.1	43.0	-	-	67.6	43.5	-	-	61.2	42.2	-	-				
	Feed		100.0	33.1	100.0	33.1	100.0	32.9	100.0	32.9	100.0	35.7	100.0	35.7	100.0	33.3	100.0	33.3				
BH8	Conc 1	0.50	23.0	21.2	23.0	21.2	20.5	20.1	20.5	20.1	23.4	20.3	23.4	20.3								
	Conc 2	1.00	11.3	20.1	34.2	20.8	9.1	18.7	29.6	19.7	11.7	19.3	35.1	20.0								
	Conc 3	2.00	13.9	19.5	48.1	20.5	20.1	18.7	49.7	19.3	11.8	18.5	46.9	19.6								
	Conc 4	4.00	14.0	18.0	62.1	19.9	15.3	17.9	65.0	19.0	10.1	19.7	57.0	19.6								
	Tails	4.00	37.9	37.6	-	-	35.0	42.5	-	-	43.0	44.9	-	-								
	Feed		100.0	26.6	100.0	26.6	100.0	27.2	100.0	27.2	100.0	30.5	100.0	30.5								
BH9	Conc 1	0.50	17.4	19.1	17.4	19.1	16.8	18.6	16.8	18.6	13.4	18.6	13.4	18.6								
	Conc 2	1.00	9.9	18.1	27.3	18.7	11.1	18.3	27.8	18.5	7.0	18.0	20.4	18.4								
	Conc 3	2.00	16.5	16.9	43.8	18.0	15.7	17.4	43.6	18.1	13.6	18.2	33.9	18.3								
	Conc 4	4.00	11.8	15.6	55.6	17.5	16.3	16.3	59.9	17.6	10.8	19.2	44.7	18.5								
	Tails	4.00	44.4	48.1	-	-	40.1	44.2	-	-	55.3	44.3	-	-								
	Feed		100.0	31.1	100.0	31.1	100.0	28.3	100.0	28.3	100.0	32.8	100.0	32.8								
BH10	Conc 1	0.50	16.7	18.7	16.7	18.7	18.2	18.2	18.2	18.2	9.3	19.1	9.3	19.1	16.8	19.9	16.8	19.9				
	Conc 2	1.00	5.8	17.7	22.5	18.4	7.6	17.4	25.7	18.0	7.5	18.9	16.8	19.0	5.4	19.2	22.2	19.7				
	Conc 3	2.00	9.5	18.2	32.0	18.4	7.7	18.0	33.4	18.0	14.4	19.6	31.1	19.3	5.6	19.8	27.7	19.7				
	Conc 4	4.00	7.0	20.2	38.9	18.7	6.6	19.1	40.1	18.2	13.4	21.5	44.6	20.0	3.8	21.0	31.6	19.9				
	Tails	4.00	61.1	44.5	-	-	59.9	42.6	-	-	55.4	32.6	-	-	68.4	51.7	-	-				
	Feed		100.0	34.5	100.0	34.5	100.0	32.8	100.0	32.8	100.0	27.0	100.0	27.0	100.0	41.6	100.0	41.6				

Appendix 2: Discard Dump Borehole Results

Table A2.8: Flotation Rate Test Results Continued

Sample	Stream	Time (min)	Sample 1				Sample 2				Sample 3				Sample 4				Sample 5			
			Fractional		Cumulative		Fractional		Cumulative		Fractional		Cumulative		Fractional		Cumulative		Fractional		Cumulative	
			Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)
BH11	Conc 1	0.50	11.0	19.3	11.0	19.3	13.5	19.7	13.5	19.7	16.5	20.5	16.5	20.5	11.0	19.1	11.0	19.1				
	Conc 2	1.00	10.1	19.8	21.1	19.5	10.2	19.2	23.7	19.5	8.3	20.1	24.8	20.4	9.1	19.1	20.0	19.1				
	Conc 3	2.00	13.2	20.9	34.4	20.1	12.0	20.2	35.7	19.7	10.4	20.5	35.2	20.4	10.9	20.3	30.9	19.5				
	Conc 4	4.00	10.0	23.3	44.4	20.8	8.8	21.7	44.5	20.1	8.8	22.3	44.0	20.8	7.0	21.3	37.9	19.9				
	Tails	4.00	55.6	36.2	-	-	55.5	46.0	-	-	56.0	43.8	-	-	62.1	42.5	-	-				
	Feed		100.0	29.4	100.0	29.4	100.0	34.5	100.0	34.5	100.0	33.7	100.0	33.7	100.0	33.9	100.0	33.9				
BH12	Conc 1	0.50	17.2	19.8	17.2	19.8	15.2	22.9	15.2	22.9	13.7	19.5	13.7	19.5	22.6	18.2	22.6	18.2				
	Conc 2	1.00	11.1	19.2	28.3	19.6	8.9	22.0	24.1	22.6	11.5	19.4	25.2	19.5	7.9	17.8	30.5	18.1				
	Conc 3	2.00	9.6	20.1	37.9	19.7	9.7	22.5	33.8	22.5	14.5	20.4	39.7	19.8	8.4	18.2	38.9	18.1				
	Conc 4	4.00	7.6	21.8	45.5	20.1	6.7	25.0	40.5	23.0	5.6	22.2	45.3	20.1	6.1	19.7	45.0	18.3				
	Tails	4.00	54.5	44.7	-	-	59.5	44.3	-	-	54.7	41.5	-	-	55.0	41.2	-	-				
	Feed		100.0	33.5	100.0	33.5	100.0	35.7	100.0	35.7	100.0	31.8	100.0	31.8	100.0	30.9	100.0	30.9				
BH13	Conc 1	0.50	25.5	19.3	25.5	19.3	12.9	17.5	12.9	17.5	21.0	18.5	21.0	18.5	18.9	20.5	18.9	20.5				
	Conc 2	1.00	9.6	17.8	35.1	18.9	10.4	17.5	23.3	17.5	8.8	17.5	29.9	18.2	8.8	19.8	27.7	20.3				
	Conc 3	2.00	10.9	18.1	45.9	18.7	12.5	17.1	35.8	17.4	9.2	17.7	39.0	18.1	9.0	20.0	36.7	20.2				
	Conc 4	4.00	7.4	20.8	53.4	19.0	10.5	18.9	46.3	17.7	6.5	19.1	45.6	18.2	6.9	21.5	43.6	20.4				
	Tails	4.00	46.6	49.3	-	-	53.7	45.9	-	-	54.4	37.5	-	-	56.4	45.6	-	-				
	Feed		100.0	33.2	100.0	33.2	100.0	32.8	100.0	32.8	100.0	29.1	100.0	29.1	100.0	34.6	100.0	34.6				
BH14	Conc 1	0.50	17.6	17.5	17.6	17.5	24.2	18.5	24.2	18.5	22.5	18.5	22.5	18.5	19.9	19.7	19.9	19.7				
	Conc 2	1.00	12.4	17.2	30.0	17.4	7.2	17.9	31.5	18.4	8.1	17.6	30.6	18.3	6.9	18.7	26.8	19.4				
	Conc 3	2.00	10.6	17.1	40.6	17.3	10.0	18.6	41.5	18.4	8.9	18.0	39.5	18.2	6.8	19.5	33.6	19.5				
	Conc 4	4.00	8.3	18.8	48.9	17.6	5.7	20.4	47.2	18.7	6.2	19.6	45.8	18.4	5.2	21.6	38.8	19.7				
	Tails	4.00	51.1	47.9	-	-	52.8	46.5	-	-	54.2	49.1	-	-	61.2	49.7	-	-				
	Feed		100.0	33.1	100.0	33.1	100.0	33.4	100.0	33.4	100.0	35.0	100.0	35.0	100.0	38.1	100.0	38.1				
BH15	Conc 1	0.50	15.8	19.0	15.8	19.0	18.5	18.4	18.5	18.4	27.4	19.1	27.4	19.1	24.9	21.3	24.9	21.3				
	Conc 2	1.00	7.2	17.7	23.0	18.6	11.3	18.1	29.8	18.3	8.5	18.3	35.9	18.9	3.6	19.9	28.5	21.1				
	Conc 3	2.00	8.0	17.1	30.9	18.2	9.5	17.9	39.3	18.2	8.3	18.6	44.2	18.9	8.3	20.9	36.7	21.1				
	Conc 4	4.00	14.1	17.7	45.1	18.1	9.7	19.6	49.0	18.5	5.9	20.7	50.1	19.1	6.0	23.2	42.7	21.4				
	Tails	4.00	54.9	44.7	-	-	51.0	45.2	-	-	49.9	49.8	-	-	57.3	46.7	-	-				
	Feed		100.0	32.7	100.0	32.7	100.0	32.1	100.0	32.1	100.0	34.4	100.0	34.4	100.0	35.9	100.0	35.9				
BH16	Conc 1	0.50	20.2	18.8	20.2	18.8	25.4	18.1	25.4	18.1	26.6	18.0	26.6	18.0	21.6	18.3	21.6	18.3				
	Conc 2	1.00	7.7	17.6	27.9	18.5	8.8	17.3	34.2	17.9	12.0	17.6	38.6	17.9	11.7	17.6	33.3	18.1				
	Conc 3	2.00	9.5	17.5	37.5	18.2	12.0	17.4	46.1	17.8	11.7	17.1	50.3	17.7	10.5	17.3	43.8	17.9				
	Conc 4	4.00	8.6	19.0	46.1	18.4	7.6	18.5	53.7	17.9	7.5	18.2	57.8	17.8	6.5	18.7	50.3	18.0				
	Tails	4.00	53.9	46.2	-	-	46.3	50.2	-	-	42.2	53.6	-	-	49.7	47.8	-	-				
	Feed		100.0	33.4	100.0	33.4	100.0	32.8	100.0	32.8	100.0	32.9	100.0	32.9	100.0	32.8	100.0	32.8				
BH17	Conc 1	0.50	23.8	16.8	23.8	16.8	25.8	17.6	25.8	17.6	22.1	18.7	22.1	18.7	17.7	18.6	17.7	18.6				
	Conc 2	1.00	8.0	16.1	31.8	16.6	10.8	16.8	36.6	17.4	6.3	19.0	28.5	18.8	3.0	17.7	20.7	18.5				
	Conc 3	2.00	7.6	17.0	39.4	16.7	10.1	16.9	46.7	17.3	6.3	19.1	34.8	18.8	6.3	19.4	27.0	18.7				
	Conc 4	4.00	4.9	19.4	44.3	17.0	6.7	18.5	53.3	17.4	5.3	20.3	40.1	19.0	4.5	20.6	31.5	19.0				
	Tails	4.00	55.7	48.3	-	-	46.7	50.6	-	-	59.9	44.2	-	-	68.5	43.3	-	-				
	Feed		100.0	34.4	100.0	34.4	100.0	32.9	100.0	32.9	100.0	34.1	100.0	34.1	100.0	35.6	100.0	35.6				
BH18	Conc 1	0.50	23.9	18.1	23.9	18.1	24.1	18.0	24.1	18.0	24.6	19.6	24.6	19.6	27.3	20.3	27.3	20.3				
	Conc 2	1.00	9.3	17.4	33.1	17.9	9.9	17.9	34.0	18.0	8.4	19.0	33.0	19.4	8.0	19.5	35.4	20.1				
	Conc 3	2.00	8.4	17.9	41.6	17.9	9.2	18.5	43.1	18.1	8.2	19.6	41.2	19.5	6.9	20.5	42.3	20.2				
	Conc 4	4.00	5.5	19.6	47.0	18.1	4.6	20.8	47.7	18.4	4.7	21.5	45.9	19.7	4.5	22.6	46.8	20.4				
	Tails	4.00	53.0	47.1	-	-	52.3	45.8	-	-	54.1	48.9	-	-	53.2	49.1	-	-				
	Feed		100.0	33.4	100.0	33.4	100.0	32.7	100.0	32.7	100.0	35.5	100.0	35.5	100.0	35.7	100.0	35.7				
BH19	Conc 1	0.50	26.3	17.7	26.3	17.7	31.1	19.0	31.1	19.0	21.7	18.3	21.7	18.3	24.2	19.3	24.2	19.3				
	Conc 2	1.00	9.1	17.5	35.4	17.6	10.6	18.6	41.7	18.9	11.4	19.0	33.1	18.5	8.6	18.5	32.8	19.1				
	Conc 3	2.00	7.3	18.4	42.7	17.8	8.5	18.9	50.1	18.9	9.2	19.2	42.2	18.7	7.8	18.8	40.6	19.0				
	Conc 4	4.00	4.7	20.4	47.4	18.0	5.4	21.2	55.5	19.1	6.4	21.4	48.6	19.0	5.3	20.8	46.0	19.2				
	Tails	4.00	52.6	46.5	-	-	44.5	50.0	-	-	51.4	50.1	-	-	54.0	44.2	-	-				
	Feed		100.0	33.0	100.0	33.0	100.0	32.8	100.0	32.8	100.0	35.0	100.0	35.0	100.0	32.7	100.0	32.7				
BH20	Conc 1	0.50	28.0	19.8	28.0	19.8	16.0	20.4	16.0	20.4	21.7	20.2	21.7	20.2	22.6	16.3	22.6	16.3				
	Conc 2	1.00	9.5	18.8	37.5	19.5	6.1	20.1	22.1	20.3	9.0	19.3	30.7	19.9	7.4	16.1	29.9	16.3				
	Conc 3	2.00	7.4	19.5	44.9	19.5	6.5	20.1	28.6	20.3	8.0	20.0	38.7	19.9	6.9	16.8	36.9	16.4				
	Conc 4	4.00	5.0	21.7	49.9	19.8	5.6	20.5	34.2	20.3	6.1	21.5	44.8	20.2	5.3	18.5	42.1	16.6				
	Tails	4.00	50.1	47.7	-	-	65.8	41.4	-	-	55.2	45.4	-	-	57.9	43.7	-	-				
	Feed		100.0	33.8	100.0	33.8	100.0	34.2	100.0	34.2	100.0	34.1	100.0	34.1	100.0	32.3	100.0	32.3				

Appendix 2: Discard Dump Borehole Results

Table A2.8: Flotation Rate Test Results Continued

Sample	Stream	Time (min)	Sample 1				Sample 2				Sample 3				Sample 4				Sample 5			
			Fractional		Cumulative		Fractional		Cumulative		Fractional		Cumulative		Fractional		Cumulative		Fractional		Cumulative	
			Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)	Yield (%)	Ash (dry %)
BH21	Conc 1	0.50	24.7	17.8	24.7	17.8	21.4	18.2	21.4	18.2	33.8	19.9	33.8	19.9	18.8	21.2	18.8	21.2				
	Conc 2	1.00	8.7	17.1	33.5	17.6	8.6	17.4	30.0	18.0	10.5	19.7	44.3	19.9	10.2	20.6	29.0	21.0				
	Conc 3	2.00	9.0	17.7	42.5	17.6	8.3	18.3	38.4	18.0	7.8	20.4	52.1	19.9	15.2	20.8	44.2	20.9				
	Conc 4	4.00	6.1	19.6	48.6	17.9	5.5	20.1	43.9	18.3	5.6	22.9	57.7	20.2	5.0	23.1	49.3	21.1				
	Tails	4.00	51.4	45.2	-	-	56.1	45.7	-	-	42.3	54.0	-	-	50.7	44.7	-	-				
	Feed		100.0	31.9	100.0	31.9	100.0	33.7	100.0	33.7	100.0	34.5	100.0	34.5	100.0	33.2	100.0	33.2				
BH22	Conc 1	0.50	32.2	21.5	32.2	21.5	25.9	17.8	25.9	17.8	27.5	19.8	27.5	19.8								
	Conc 2	1.00	10.3	21.1	42.4	21.4	8.0	17.3	33.8	17.7	11.4	19.4	38.9	19.7								
	Conc 3	2.00	9.6	20.9	52.1	21.3	8.3	18.2	42.2	17.8	10.4	19.2	49.3	19.6								
	Conc 4	4.00	5.2	22.9	57.3	21.5	6.1	20.3	48.3	18.1	7.1	21.0	56.4	19.8								
	Tails	4.00	42.7	49.9	-	-	51.7	46.9	-	-	43.6	50.4	-	-								
	Feed		100.0	33.6	100.0	33.6	100.0	33.0	100.0	33.0	100.0	33.1	100.0	33.1								
BH23	Conc 1	0.50	23.2	19.2	23.2	19.2	19.3	19.7	19.3	19.7	15.9	18.3	15.9	18.3	19.6	18.9	19.6	18.9				
	Conc 2	1.00	7.5	18.3	30.7	19.0	7.1	18.5	26.4	19.4	6.3	17.8	22.2	18.2	5.7	18.0	25.3	18.7				
	Conc 3	2.00	7.0	18.4	37.7	18.9	7.7	18.9	34.2	19.3	7.1	18.0	29.3	18.1	12.6	18.9	37.9	18.8				
	Conc 4	4.00	5.6	19.4	43.2	18.9	6.9	20.0	41.1	19.4	7.0	18.7	36.3	18.2	7.8	22.2	45.7	19.4				
	Tails	4.00	56.8	45.1	-	-	58.9	41.7	-	-	63.7	41.1	-	-	54.3	43.9	-	-				
	Feed		100.0	33.8	100.0	33.8	100.0	32.6	100.0	32.6	100.0	32.8	100.0	32.8	100.0	32.5	100.0	32.5				
BH24	Conc 1	0.50	20.4	19.6	20.4	19.6	16.8	19.0	16.8	19.0	11.6	20.0	11.6	20.0	7.0	20.5	7.0	20.5				
	Conc 2	1.00	8.0	19.4	28.3	19.5	4.5	18.1	21.3	18.8	11.1	20.4	22.7	20.2	4.1	19.8	11.1	20.2				
	Conc 3	2.00	8.2	20.0	36.6	19.6	11.2	18.4	32.5	18.7	7.4	19.9	30.1	20.1	8.6	19.9	19.8	20.1				
	Conc 4	4.00	6.0	21.6	42.5	19.9	9.0	20.1	41.5	19.0	11.1	20.6	41.2	20.3	9.5	21.4	29.2	20.5				
	Tails	4.00	57.5	47.4	-	-	58.5	45.1	-	-	58.8	52.1	-	-	70.8	48.1	-	-				
	Feed		100.0	35.7	100.0	35.7	100.0	34.3	100.0	34.3	100.0	39.0	100.0	39.0	100.0	40.1	100.0	40.1				
BH25	Conc 1	0.50	27.0	20.3	27.0	20.3	8.5	20.0	8.5	20.0	21.4	21.9	21.4	21.9								
	Conc 2	1.00	9.3	20.4	36.3	20.3	10.8	20.1	19.3	20.1	8.8	21.0	30.2	21.6								
	Conc 3	2.00	8.4	21.5	44.7	20.5	11.4	20.3	30.7	20.1	8.3	22.8	38.5	21.9								
	Conc 4	4.00	4.2	24.9	48.8	20.9	8.8	21.5	39.5	20.4	5.7	24.1	44.2	22.2								
	Tails	4.00	51.2	48.9	-	-	60.5	37.4	-	-	55.8	48.4	-	-								
	Feed		100.0	35.3	100.0	35.3	100.0	30.7	100.0	30.7	100.0	36.8	100.0	36.8								
BH26	Conc 1	0.50	22.6	18.7	22.6	18.7	22.8	20.0	22.8	20.0	21.0	20.8	21.0	20.8								
	Conc 2	1.00	8.7	18.8	31.3	18.7	10.9	20.5	33.7	20.2	10.0	20.0	31.1	20.5								
	Conc 3	2.00	8.2	19.6	39.5	18.9	9.8	20.9	43.5	20.3	8.5	21.1	39.6	20.7								
	Conc 4	4.00	5.6	22.1	45.1	19.3	5.5	21.9	49.0	20.5	5.2	23.0	44.8	20.9								
	Tails	4.00	54.9	49.4	-	-	51.0	48.5	-	-	55.2	45.2	-	-								
	Feed		100.0	35.8	100.0	35.8	100.0	35.5	100.0	35.5	100.0	34.3	100.0	34.3								
BH27	Conc 1	0.50	19.0	19.4	19.0	19.4	17.2	19.0	17.2	19.0												
	Conc 2	1.00	7.0	18.6	26.0	19.2	6.1	18.3	23.3	18.8												
	Conc 3	2.00	7.1	19.1	33.1	19.2	6.5	18.8	29.8	18.8												
	Conc 4	4.00	5.6	20.6	38.6	19.4	5.1	19.9	34.9	19.0												
	Tails	4.00	61.4	45.1	-	-	65.1	44.5	-	-												
	Feed		100.0	35.1	100.0	35.1	100.0	35.6	100.0	35.6												
BH28	Conc 1	0.50	20.9	21.7	20.9	21.7	16.7	13.5	16.7	13.5												
	Conc 2	1.00	8.0	21.4	28.9	21.6	6.2	20.1	22.9	15.3												
	Conc 3	2.00	8.4	21.8	37.3	21.7	6.6	20.5	29.5	16.4												
	Conc 4	4.00	6.4	23.5	43.7	21.9	5.3	21.9	34.8	17.3												
	Tails	4.00	56.3	48.2	-	-	65.2	45.1	-	-												
	Feed		100.0	36.7	100.0	36.7	100.0	35.5	100.0	35.5												
BH29	Conc 1	0.50	23.7	18.9	23.7	18.9																
	Conc 2	1.00	17.7	17.6	41.4	18.3																
	Conc 3	2.00	14.3	16.2	55.7	17.8																
	Conc 4	4.00	20.2	15.4	75.9	17.2																
	Tails	4.00	24.1	47.5	-	-																
	Feed		100.0	24.5	100.0	24.5																
BH30	Conc 1	0.50	21.8	17.9	21.8	17.9																
	Conc 2	1.00	14.0	16.6	35.8	17.4																
	Conc 3	2.00	12.6	15.2	48.4	16.8																
	Conc 4	4.00	20.6	15.2	68.9	16.3																
	Tails	4.00	31.1	42.5	-	-																
	Feed		100.0	24.5	100.0	24.5																

Appendix 3: Current Arising Discard Monthly Composite Results

This appendix contains the data from the various tests performed on the current arising monthly discard composite samples for April 2008, May 2008, June 2008 and August 2008.

A3.1 April 2008

Table A3.1: Sample subjected to screening and destoning prior to milling and floating

Stream	Mass (%)	Ash (%)
Feed	100.0	54.7
Floats 1.80	37.2	29.2
Sinks 1.80	57.7	71.6
-0.5mm	5.4	46.3
Flotation Feed	42.6	31.4
Flotation Concentrate	31.3	23.1
Flotation Tailings	11.7	46.8

Table A3.2: Flotation rate test results

Stream	Mass Yield (%)	Cum. Yield (%)	Ash (%)	Cum. Ash (%)
Feed	100.0	0.0	29.6	0.0
Conc 1	35.3	35.3	21.5	21.5
Conc 2	24.1	59.4	24.0	22.6
Conc 3	8.7	68.1	24.3	22.8
Conc 4	4.3	72.4	28.6	23.1
Tails	27.6	100.0	46.8	29.6

Table A3.3: Sample of the -1.4mm fraction of the discard floated “as is”

Stream	Mass Yield (%)	Cum. Yield (%)	Ash (%)	Cum. Ash (%)
Feed		0.0	38.4	0.0
Conc 1	8.4	8.4	18.0	18.0
Conc 2	7.3	15.7	25.0	21.3
Conc 3	5.2	20.9	49.7	28.4
Conc 4	10.0	30.9	52.8	36.3
Tails	69.1	100.0	52.2	47.3

A3.2 May 2008

Table A3.4: Sample subjected to screening and destoning prior to milling and floating

Stream	Mass (%)	Ash (%)
Feed	100.0	57.7
Floats 1.80	33.2	30.3
Sinks 1.80	65.4	71.9
-0.5mm	1.4	43
Flotation Feed	34.6	30.8
Flotation Concentrate	18.5	20.0
Flotation Tailings	16.1	43.1

Table A3.5: Flotation rate test results

Stream	Mass Yield (%)	Cum. Yield (%)	Ash (%)	Cum. Ash (%)
Feed	100.0	0.0	30.7	0.0
Conc 1	24.2	24.2	19.4	19.4
Conc 2	11.6	35.8	19.2	19.4
Conc 3	10.4	46.2	20.3	19.6
Conc 4	7.3	53.5	22.7	20.0
Tails	46.5	100.0	43.1	30.7

Table A3.6: Sample of the -1.4mm fraction of the discard floated “as is”

Stream	Mass Yield (%)	Cum. Yield (%)	Ash (%)	Cum. Ash (%)
Feed		0.0	37.3	0.0
Conc 1	11.6	11.6	14.3	14.3
Conc 2	9.0	20.5	17.2	15.6
Conc 3	1.2	21.7	33.8	16.6
Conc 4	0.8	22.5	44.1	17.5
Tails	71.5	94.0	45.2	38.6

A4.3 June 2008

This sample (and the August 2008) was treated slightly different to the two previous samples in that it was screened to give a detailed particle size distribution. The +13mm (+12.5mm) size fractions were crushed to -13mm prior to the destoning, milling and flotation process.

Table A3.7: Particle size distribution results

Size Fraction (mm)	% Retained		Cumulative Passing size (%)
	Fractional	Cumulative	
+20	10.2	10.2	89.8
-20+18	2.6	12.8	87.2
-18+12.5	13.5	26.4	73.6
-12.5+8	17.8	44.2	55.8
-8+3	24.0	68.2	31.8
-3+1.4	14.7	82.9	17.1
-1.4+0.6	12.1	95.0	5.0
-0.6+0.212	3.3	98.3	1.7
-0.212+0.125	0.7	99.0	1.0
-0.125+0.075	0.5	99.5	0.5
-0.075	0.0	100.0	0.0
Weighted Mean Size	8.2		

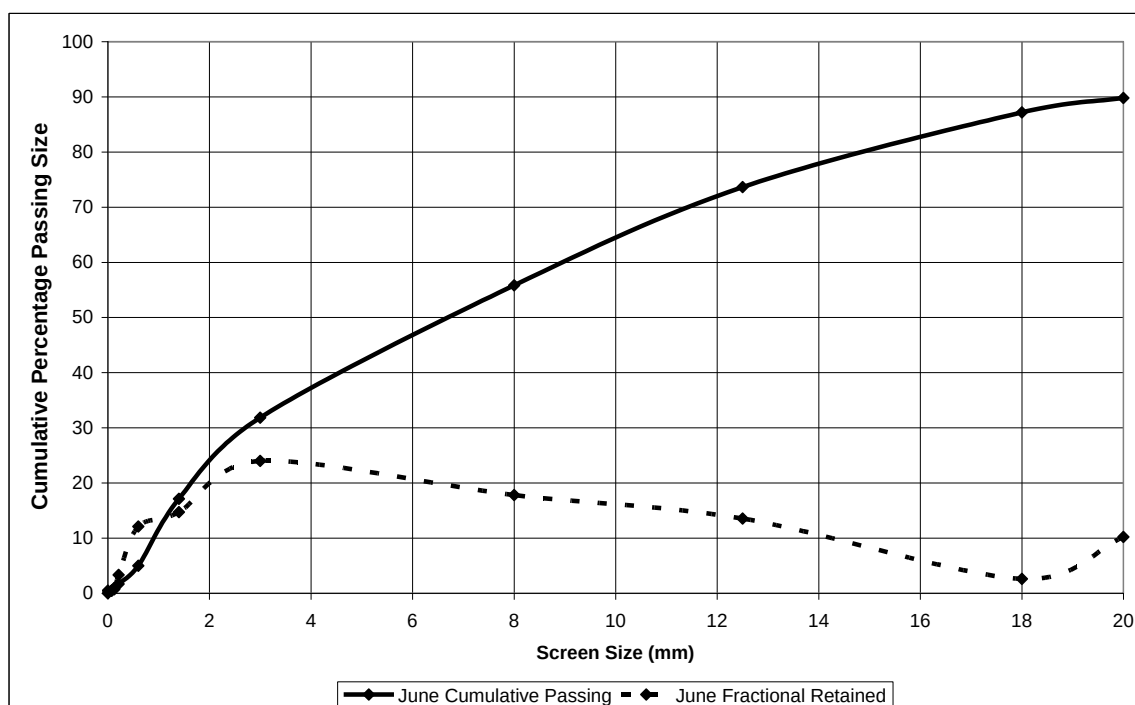


Figure A3.1: Particle size distribution results

Table A3.8: Sample subjected to screening and destoning prior to milling and floating

Stream	Mass (%)	Ash (%)
Feed	100.0	55.3
Floats 1.80	34.8	28.0
Sinks 1.80	60.2	71.6
-0.5mm	5.0	47.8
Flotation Feed	39.8	30.5
Flotation Concentrate	22.4	19.81
Flotation Tailings	17.4	45.04

Table A3.9: Flotation rate test results

Stream	Mass Yield (%)	Cum. Yield (%)	Ash (%)	Cum. Ash (%)
Feed	100.0	0.0	30.7	0.0
Conc 1	18.6	18.6	19.2	19.2
Conc 2	15.6	34.3	19.1	19.2
Conc 3	13.2	47.4	19.3	19.2
Conc 4	8.9	56.3	21.9	19.6
Tails	43.7	100.0	45.0	30.7

Two additional flotation rate tests were also performed on the June 2008 (and August 2008) sample. The first being performed on the sample ‘as is” (i.e. not milling prior to flotation) on the -1.4mm material to examine the effect of either refloating this fraction directly from the plant or extending the flotation residence time in the plant and additional reagent additions. The results are presented in Table A3.10 below.

Table A3.10: Sample of the -1.4mm fraction of the discard floated “as is”

Stream	Mass Yield (%)	Cum. Yield (%)	Ash (%)	Cum. Ash (%)
Feed		0.0	38.6	0.0
Conc 1	15.8	15.8	14.9	14.9
Conc 2	7.1	22.9	22.9	17.4
Conc 3	1.6	24.5	31.9	18.3
Conc 4	0.4	24.9	32.6	18.6
Tails	75.1	100.0	45.3	38.6

The second test examined the effect of floating the -1.4mm material after milling to -212µm without destoning of the +0.5mm fraction. The results are summarised in Table A3.11.

Table A3.11: Sample of the -1.4mm fraction of the discard milled to -212µm prior to flotation

Stream	Mass Yield (%)	Cum. Yield (%)	Ash (%)	Cum. Ash (%)
Feed	100.0	0.0	38.2	0.0
Conc 1	29.9	29.9	17.4	17.4
Conc 2	10.1	40.0	19.0	17.8
Conc 3	7.3	47.3	20.7	18.2
Conc 4	5.0	52.4	24.1	18.8
Tails	47.6	100.0	59.6	38.2

A3.4 August 2008

The August 2008 sample was also treated in a manner similar to that of June 2008 in terms of the determination of the particle size distribution and additional flotation rate tests.

Table A3.12: Particle size distribution results

Size Fraction (mm)	% Retained		Cumulative Passing size (%)
	Fractional	Cumulative	
+20	15.5	15.5	84.5
-20+18	4.4	19.9	80.1
-18+12.5	13.8	33.7	66.3
-12.5+8	15.1	48.7	51.3
-8+3	20.1	68.8	31.2
-3+1.4	13.5	82.4	17.6
-1.4+0.6	12.0	94.4	5.6
-0.6+0.212	3.1	97.5	2.5
-0.212+0.125	0.4	97.9	2.1
-0.125+0.075	1.7	99.6	0.4
-0.075	0.4	100.0	0.0
Weighted Mean Size	9.1		

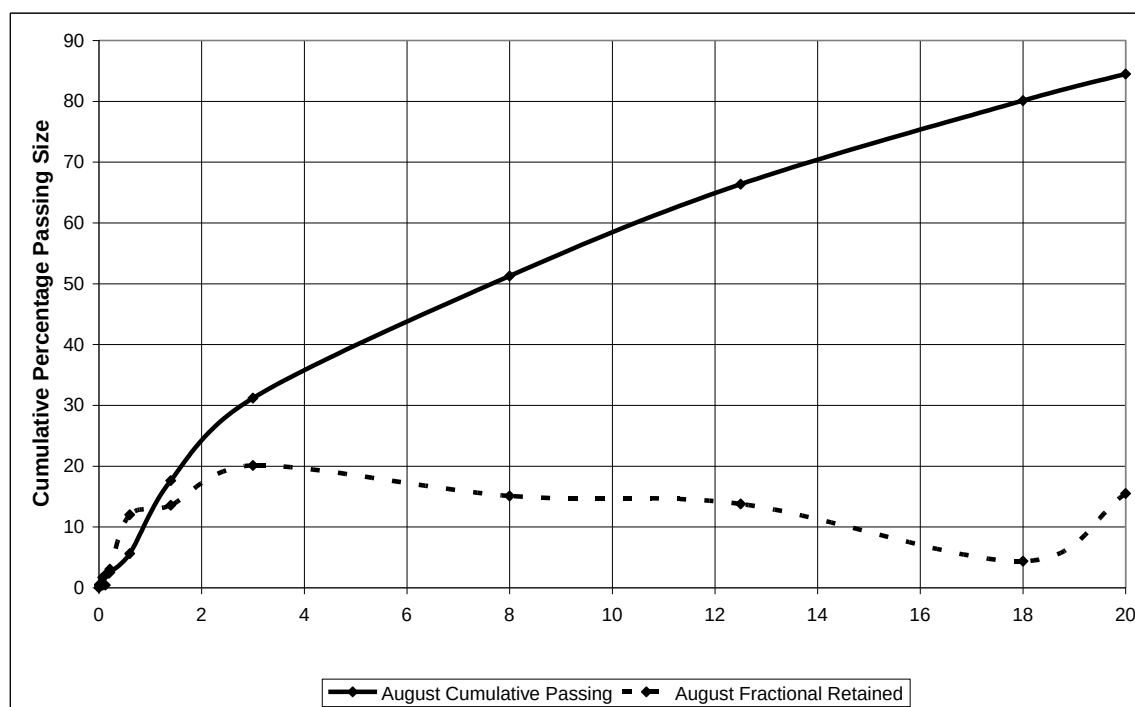


Figure A3.2: Particle size distribution results

Table A3.13: Sample subjected to screening and destoning prior to milling and floating

Stream	Mass (%)	Ash (%)
Feed	100.0	57.6
Floats 1.80	31.5	27.6
Sinks 1.80	62.9	73.1
-0.5mm	5.6	53.2
Flotation Feed	37.1	31.5
Flotation Concentrate	25.9	21.3
Flotation Tailings	11.2	49.8

Table A3.14: Flotation rate test results

Stream	Mass Yield (%)	Cum. Yield (%)	Ash (%)	Cum. Ash (%)
Feed	100.0	0.0	43.6	0.0
Conc 1	29.4	29.4	17.8	17.8
Conc 2	10.4	39.8	18.5	18.0
Conc 3	7.0	46.8	22.8	18.7
Conc 4	2.6	49.4	29.8	19.3
Tails	50.6	100.0	67.3	43.6

Table A3.15: Sample of the -1.4mm fraction of the discard floated “as is”

Stream	Mass Yield (%)	Cum. Yield (%)	Ash (%)	Cum. Ash (%)
Feed		0.0	40.9	0.0
Conc 1	12.8	12.8	14.6	14.6
Conc 2	8.4	21.2	24.8	18.6
Conc 3	3.6	24.8	34.9	21.0
Conc 4	4.4	29.3	37.9	23.5
Tails	70.7	100.0	48.1	40.9

Table A3.16: Sample of the -1.4mm fraction of the discard milled to -212µm prior to flotation

Stream	Mass Yield (%)	Cum. Yield (%)	Ash (%)	Cum. Ash (%)
Feed	100.0	0.0	26.8	0.0
Conc 1	31.5	31.5	15.6	15.6
Conc 2	9.0	40.5	15.0	15.4
Conc 3	4.7	45.2	16.0	15.5
Conc 4	4.5	49.7	18.8	15.8
Tails	50.3	100.0	37.6	26.8

Appendix 4: Current Arising Discard Bulk Sample Results

This appendix, which relates to the work discussed in Chapter 4, contains the data from the various tests performed on the current arising discard samples from:

- 2 samples from Mutale Shaft, each named differently to differentiate them
 - Mutale Shaft
 - Mutale Plant Waste
- 2 samples from Vhukati Shaft, each named differently to differentiate them
 - Vhukati Shaft
 - Vhukati Plant Waste
- 1 sample from Goni Shaft
 - Goni Plant Waste

Each of the samples was screened into suitably selected size classes and each size class was then subjected to float and sinks testing. Thereafter the data was used for modelling purposes to determine the mass balance for a proposed destoning plant.

A4.1 Mutale Shaft

A4.1.1 Particle Size Distribution

The particle size distribution of the Mutale Shaft discard sample was determined prior to any further processing and is presented in Table A4.1 and Figure A4.1.

Table A4.1: Particle size distribution of the sample as produced by the plant

Size (mm)	Mass (g)	Mass (%)	Passing (%)
20			100.00
16	26420	26.27	73.73
8	27420	27.26	46.47
4	14531	14.45	32.02
2	9350	9.30	22.72
1	12794	12.72	10.00
0.5	6764	6.73	3.28
0	3294	3.28	0
Total	100573	100.00	

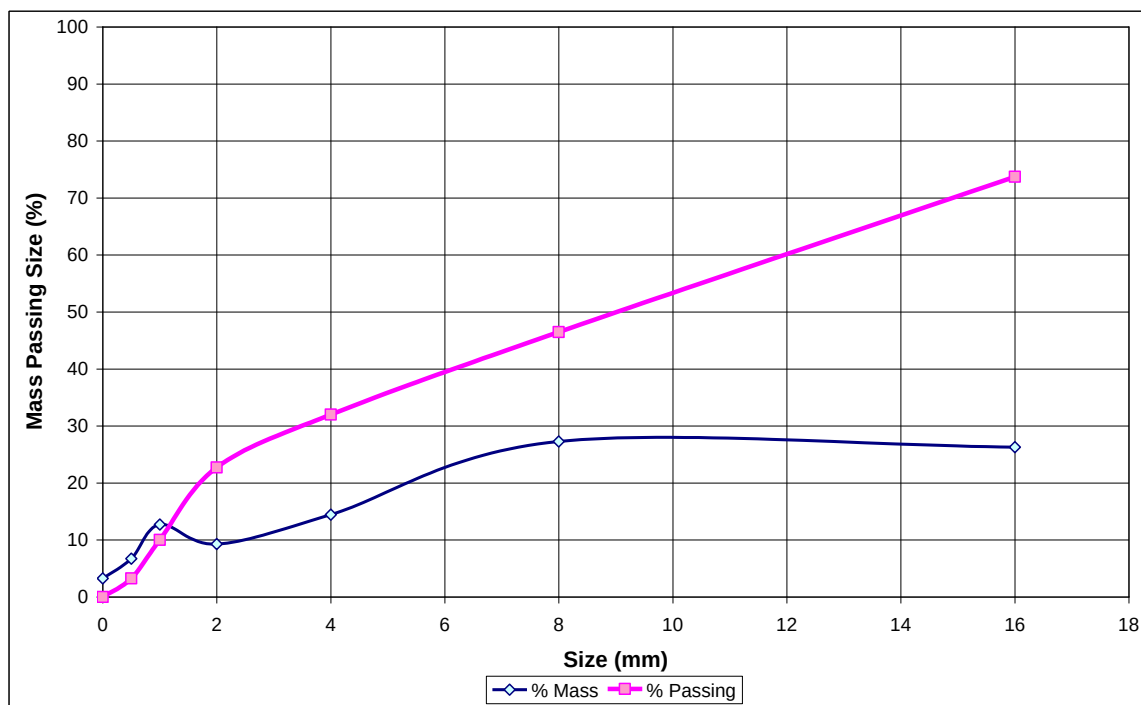


Figure A4.1: Particle size distribution of the sample as produced by the plant

A4.1.2 Mutale Shaft Screening Separation Simulations

Having established the particle size distribution, each size fraction was analyzed for ash content and it was evident that 8mm would be a potentially suitable size to remove coarse high ash material and allow for destoning to take place. Two simulations were therefore performed using a d_{50} of 8mm and Ep values of 0.200 and 0.250 respectively. These are summarized in Table A4.2 and Table A4.3.

Table A4.2: Screening simulation at a d_{50} of 8mm and Ep of 0.200

Size Fraction (mm)		Mean	Input Data		Simulated	Screen Oversize		Screen Undersize	
Upper	Lower	Size (mm)	Yield	Ash	Partition Number	Yield (%)	Ash (%)	Yield (%)	Ash (%)
20.0	16.00	17.89	27.8	75.7	1.000	27.8	75.7	0.0	75.7
16.0	8.00	11.31	29.6	62.9	1.000	29.6	62.9	0.0	62.9
8.0	4.00	5.66	16.2	58.5	0.000	0.0	58.5	16.2	58.5
4.0	2.00	2.83	10.4	54.7	0.000	0.0	54.7	10.4	54.7
2.0	1.00	1.41	10.1	39.1	0.000	0.0	39.1	10.1	39.1
1.0	0.50	0.71	3.6	39.1	0.000	0.0	39.1	3.6	39.1
0.5	0.00	0.01	2.2	29.3	0.000	0.0	29.3	2.2	29.3
Total			100.0	60.9		57.5	69.1	42.5	49.8

Table A4.3: Screening simulation at a d_{50} of 8mm and Ep of 0.250

Size Fraction (mm)		Mean	Input Data		Simulated	Screen Oversize		Screen Undersize	
Upper	Lower	Size (mm)	Yield	Ash	Partition Number	Yield (%)	Ash (%)	Yield (%)	Ash (%)
20.0	16.00	17.89	27.8	75.7	1.000	27.8	75.7	0.0	75.7
16.0	8.00	11.31	29.6	62.9	1.000	29.6	62.9	0.0	62.9
8.0	4.00	5.66	16.2	58.5	0.000	0.0	58.5	16.2	58.5
4.0	2.00	2.83	10.4	54.7	0.000	0.0	54.7	10.4	54.7
2.0	1.00	1.41	10.1	39.1	0.000	0.0	39.1	10.1	39.1
1.0	0.50	0.71	3.6	39.1	0.000	0.0	39.1	3.6	39.1
0.5	0.00	0.01	2.2	29.3	0.000	0.0	29.3	2.2	29.3
Total			100.0	60.9		57.5	69.1	42.5	49.8

A4.1.3 Mutale Shaft Raw Washability Data by Size Fraction

Each size class was also subjected to a washability analysis, to determine the effect of destoning by dense medium separation, as detailed in Table A4.4 to Table A4.9 below:

Table A4.4: Washability data of the +16mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
+ 16 mm	Float 1.30	0.0					
	Float 1.35	0.0					
	Float 1.40	0.0					
	Float 1.45	0.1	0.9	24.1	24.3	0.1	24.3
	Float 1.50	1.1	0.7	25.4	25.6	1.1	25.5
	Float 1.55	2.9	0.7	28.0	28.2	4.0	27.4
	Float 1.60	1.4	0.6	32.1	32.3	5.4	28.7
	Float 1.65	2.4	0.6	35.6	35.9	7.9	30.9
	Float 1.70	2.3	0.7	40.1	40.3	10.1	33.0
	Float 1.75	2.5	0.7	43.6	43.9	12.6	35.2
	Float 1.80	2.2	0.7	47.6	47.9	14.8	37.0
	Float 1.85	2.3	0.7	51.4	51.8	17.1	39.0
	Float 1.90	2.5	0.7	54.2	54.6	19.6	41.0
	Sink 1.90	80.4	0.9	76.2	76.9	-	-
	Total	100.0	0.8	69.3	69.8	100.0	69.8

Table A4.5: Washability data of the -16+8mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 16 + 8 mm	Float 1.30	0.0					
	Float 1.35	0.0					
	Float 1.40	0.1	0.8	13.8	13.9	0.1	13.9
	Float 1.45	0.3	0.6	19.4	19.5	0.3	18.4
	Float 1.50	3.5	0.6	24.5	24.6	3.8	24.1
	Float 1.55	6.3	0.5	28.2	28.3	10.1	26.8
	Float 1.60	4.6	0.6	32.4	32.6	14.7	28.6
	Float 1.65	4.3	0.5	36.4	36.6	19.0	30.4
	Float 1.70	5.5	0.5	40.2	40.4	24.5	32.6
	Float 1.75	4.8	0.4	43.9	44.1	29.3	34.5
	Float 1.80	4.6	0.5	47.9	48.2	33.8	36.4
	Float 1.85	3.9	0.7	50.6	51.0	37.8	37.9
	Float 1.90	3.6	0.5	54.3	54.6	41.4	39.3
	Sink 1.90	58.6	0.8	74.6	75.2	-	-
	Total	100.0	0.7	59.9	60.3	100.0	60.3

Table A4.6: Washability data of the -8+4mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 8 + 4 mm	Float 1.30	0.0					
	Float 1.35	0.1	0.6	13.8	13.9	0.1	13.9
	Float 1.40	0.3	0.4	14.4	14.5	0.4	14.3
	Float 1.45	1.1	0.3	19.2	19.2	1.4	18.0
	Float 1.50	4.2	0.6	24.1	24.2	5.6	22.6
	Float 1.55	7.0	0.5	28.1	28.2	12.6	25.7
	Float 1.60	6.6	0.5	32.0	32.1	19.2	27.9
	Float 1.65	4.6	0.6	35.6	35.8	23.8	29.5
	Float 1.70	6.8	0.6	40.0	40.2	30.6	31.8
	Float 1.75	5.3	0.7	42.8	43.1	35.9	33.5
	Float 1.80	5.7	0.7	46.4	46.7	41.6	35.3
	Float 1.85	4.8	0.7	51.0	51.3	46.4	37.0
	Float 1.90	3.8	0.8	53.5	53.9	50.2	38.3
	Sink 1.90	49.8	0.7	73.6	74.1	-	-
	Total	100.0	0.7	55.7	56.1	100.0	56.1

Table A4.7: Washability data of the -4+2mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 4 + 2 mm	Float 1.30	0.5	0.4	4.7	4.7	0.5	4.7
	Float 1.35	0.7	0.6	8.0	8.1	1.3	6.6
	Float 1.40	1.4	0.6	12.3	12.4	2.7	9.7
	Float 1.45	2.3	0.6	18.1	18.2	5.0	13.6
	Float 1.50	4.2	0.8	22.9	23.1	9.2	18.0
	Float 1.55	7.3	0.8	27.9	28.1	16.5	22.4
	Float 1.60	6.3	0.8	31.5	31.7	22.8	25.0
	Float 1.65	4.1	0.7	36.2	36.5	26.9	26.7
	Float 1.70	6.6	0.6	39.7	40.0	33.5	29.3
	Float 1.75	5.0	0.6	43.1	43.4	38.5	31.2
	Float 1.80	5.5	0.7	48.1	48.5	44.0	33.4
	Float 1.85	3.8	0.8	53.1	53.6	47.9	35.0
	Float 1.90	3.6	0.7	55.9	56.2	51.5	36.5
	Sink 1.90	48.5	0.8	71.2	71.8	-	-
	Total	100.0	0.7	53.2	53.6	100.0	53.6

Table A4.8: Washability data of the -2+1mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 2 + 1 mm	Float 1.30	10.2	0.6	5.0	5.0	10.2	5.0
	Float 1.35	7.1	0.7	9.3	9.4	17.3	6.8
	Float 1.40	8.0	0.7	13.2	13.3	25.4	8.8
	Float 1.45	9.9	0.7	18.0	18.1	35.3	11.5
	Float 1.50	6.2	0.7	23.1	23.3	41.5	13.2
	Float 1.55	5.6	0.7	29.2	29.4	47.2	15.2
	Float 1.60	4.4	0.8	33.0	33.3	51.6	16.7
	Float 1.65	3.0	0.7	37.4	37.6	54.6	17.9
	Float 1.70	3.5	0.8	42.2	42.6	58.0	19.3
	Float 1.75	2.4	0.8	43.1	43.4	60.4	20.3
	Float 1.80	3.7	0.9	51.3	51.8	64.1	22.1
	Float 1.85	2.7	0.8	57.5	58.0	66.8	23.6
	Float 1.90	1.2	0.9	58.6	59.2	68.0	24.2
	Sink 1.90	32.0	1.2	72.1	73.0	-	-
	Total	100.0	0.9	39.4	39.8	100.0	39.8

Table A4.9: Washability data of the -1+0.5mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 1 + 0.5 mm	Float 1.30	11.4	0.7	7.4	7.5	11.4	7.5
	Float 1.35	2.7	0.7	9.5	9.6	14.1	7.9
	Float 1.40	3.6	0.6	13.0	13.1	17.7	8.9
	Float 1.45	10.9	0.8	17.8	18.0	28.6	12.4
	Float 1.50	7.8	0.8	23.0	23.1	36.4	14.7
	Float 1.55	8.3	0.8	29.3	29.5	44.7	17.4
	Float 1.60	5.2	0.8	32.8	33.0	49.9	19.1
	Float 1.65	3.1	0.7	37.3	37.6	53.0	20.1
	Float 1.70	3.6	0.8	42.2	42.5	56.6	21.6
	Float 1.75	2.8	0.7	43.3	43.7	59.4	22.6
	Float 1.80	4.6	0.8	51.5	51.9	63.9	24.7
	Float 1.85	3.7	0.9	57.8	58.3	67.6	26.5
	Float 1.90	1.0	0.9	57.7	59.2	68.6	27.0
	Sink 1.90	31.4	1.0	69.3	70.0	-	-
	Total	100.0	0.8	40.1	40.5	100.0	40.5

A4.1.4 Mutale Shaft Calculated Washability Data from Washability by Size Fraction Data

The washability data from above was then reconstituted to form a washability of the entire sample and also one for the -8+0.5mm size fraction. The latter was performed to allow for a circuit simulation utilizing both screening and dense medium separation as a method of destoning. The results are presented in Table A4.10 and Table A4.11.

Table A4.10: Reconstituted washability data of the whole discard (excluding the -0.5mm material)

Size Fraction	RD	Fractional		Cumulative	
		Yield (%)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
Total Sample	Float 1.30	3.7	6.2	3.7	6.2
	Float 1.35	1.8	9.4	5.5	7.3
	Float 1.40	2.2	13.2	7.7	9.0
	Float 1.45	4.1	18.2	11.8	12.2
	Float 1.50	4.5	23.6	16.3	15.3
	Float 1.55	6.3	28.7	22.5	19.0
	Float 1.60	4.7	32.5	27.3	21.4
	Float 1.65	3.6	36.6	30.8	23.1
	Float 1.70	4.7	40.8	35.5	25.5
	Float 1.75	3.8	43.6	39.3	27.2
	Float 1.80	4.4	49.1	43.7	29.4
	Float 1.85	3.6	53.8	47.3	31.2
	Float 1.90	2.6	55.5	49.9	32.5
	Sink 1.90	50.1	74.1	-	-
	Total	100.0	53.4	100.0	53.4

Table A4.11: Reconstituted washability data of the -8+0.5mm size fraction

Size Fraction	RD	Fractional		Cumulative	
		Yield (%)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
-8+0.5mm	Float 1.30	4.9	5.8	4.9	5.8
	Float 1.35	2.7	9.4	7.6	7.1
	Float 1.40	3.3	13.2	10.9	9.0
	Float 1.45	5.4	18.2	16.3	12.0
	Float 1.50	5.3	23.5	21.7	14.8
	Float 1.55	6.9	28.7	28.5	18.2
	Float 1.60	5.7	32.4	34.2	20.5
	Float 1.65	3.8	36.6	38.0	22.1
	Float 1.70	5.3	40.8	43.2	24.4
	Float 1.75	4.0	43.3	47.2	26.0
	Float 1.80	4.9	49.0	52.1	28.2
	Float 1.85	3.8	54.2	55.9	30.0
	Float 1.90	2.6	55.7	58.5	31.1
	Sink 1.90	41.5	72.8	-	-
	Total	100.0	48.4	100.0	48.4

A4.1.5 Mutale Shaft Screening and Dense Medium Separation Simulation

Using the reconstituted washability for the -8+0.5mm size fraction, two simulations were performed, the first to produce a standard Tshikondeni coking coal product of 14% ash, the second was for the production of an alternative product suitable for supply to Eskom at an ash content as close as possible to a typical ash specification level of 30%. The screen was assumed to cut at 8mm with an Ep of 0.250 and the dense medium separation cyclone Ep was set at 0.025, which is within industry norms.

The simulations are illustrated in Figure A4.2 and Figure A4.3.

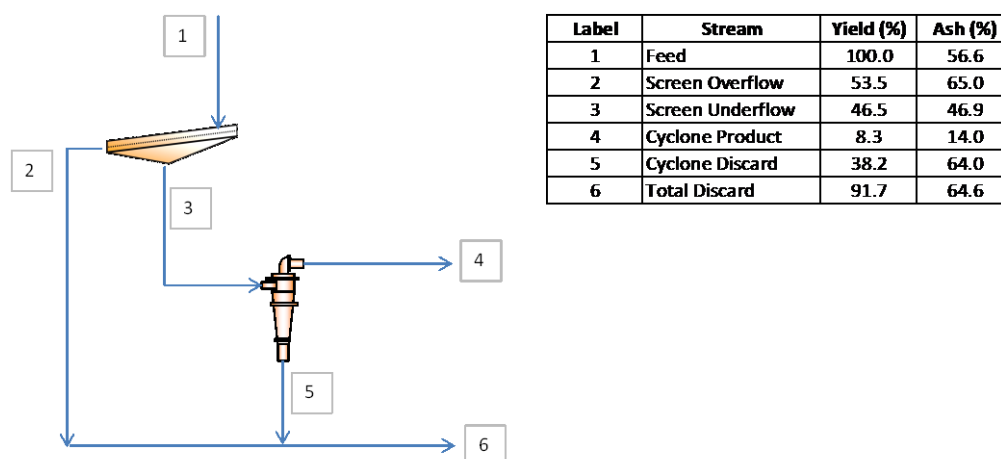


Figure A4.2: Particle size distribution of the sample as produced by the plant

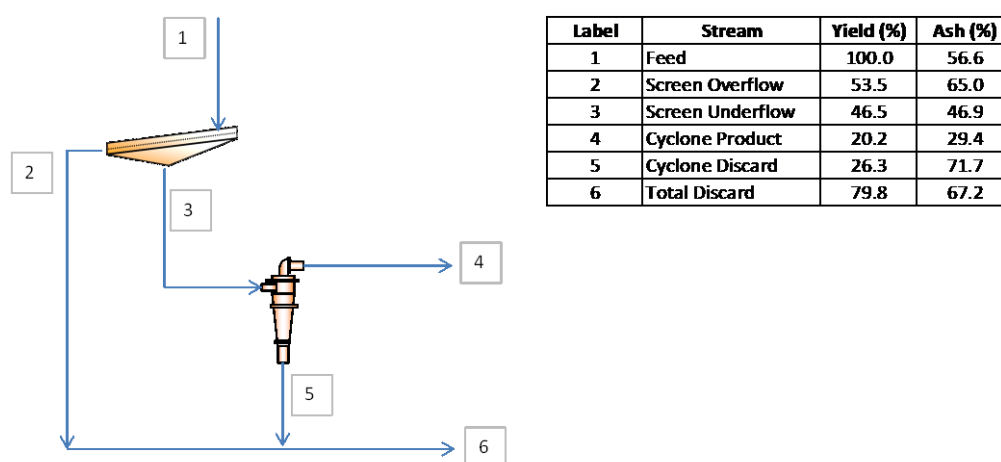


Figure A4.3: Particle size distribution of the sample as produced by the plant

A4.2 Mutale Plant Waste

A4.2.1 Particle Size Distribution

The particle size distribution of the Mutale Plant Waste discard sample was determined prior to any further processing and is presented in Table A4.12 and Figure A4.4.

Table A4.12: Particle size distribution of the sample as produced by the plant

Size (mm)	Mass (g)	Mass (%)	Passing (%)
20			100.00
16	27200	27.85	72.15
8	28940	29.63	42.52
4	15787	16.16	26.36
2	10185	10.43	15.93
1	9832	10.07	5.87
0.5	3536	3.62	2.25
0	2193	2.25	0
Total	97673	100.00	

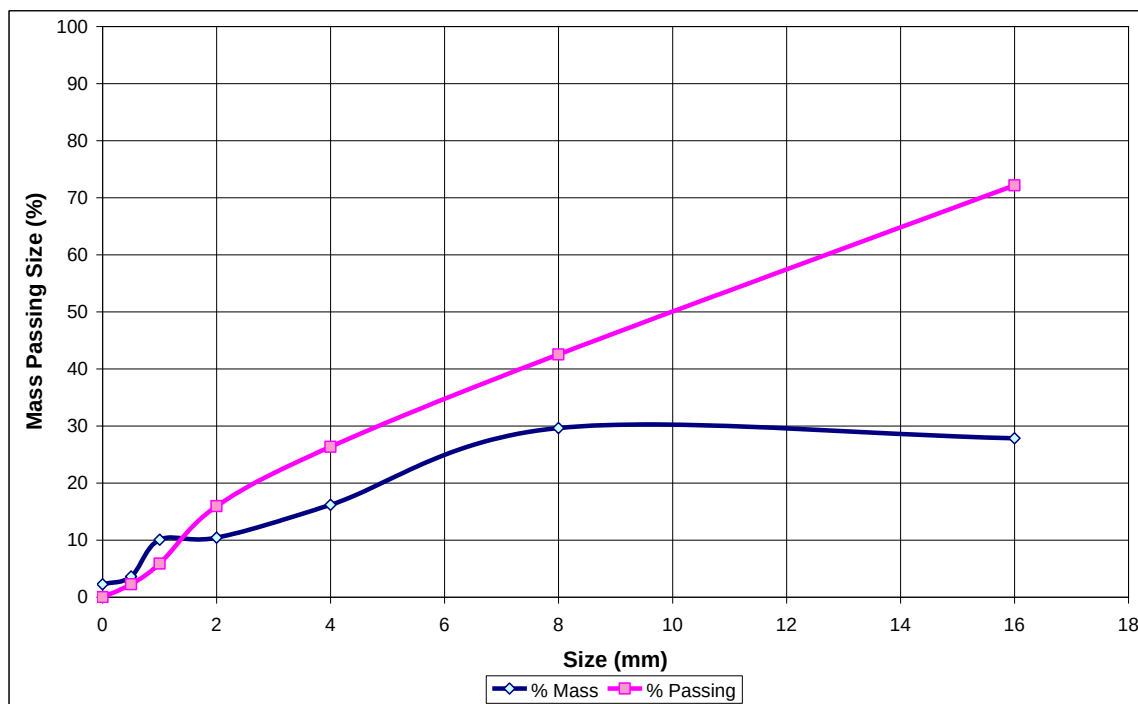


Figure A4.4: Particle size distribution of the sample as produced by the plant

A4.2.2 Mutale Plant Waste Screening Separation Simulations

Having established the particle size distribution, each size fraction was analyzed for ash content and it was evident that 8mm would be a potentially suitable size to remove coarse high ash material and allow for destoning to take place. Two simulations were therefore performed using a d_{50} of 8mm and Ep values of 0.200 and 0.250 respectively. These are summarized in Table A4.13 and Table A4.14.

Table A4.13: Screening simulation at a d_{50} of 8mm and Ep of 0.200

Size Fraction (mm)		Mean	Input Data		Simulated	Screen Oversize		Screen Undersize	
Upper	Lower	Size (mm)	Yield	Ash	Partition Number	Yield (%)	Ash (%)	Yield (%)	Ash (%)
20.0	16.00	17.89	26.3	69.8	1.000	26.3	69.8	0.0	69.8
16.0	8.00	11.31	27.3	60.3	1.000	27.3	60.3	0.0	60.3
8.0	4.00	5.66	14.4	56.1	0.000	0.0	56.1	14.4	56.1
4.0	2.00	2.83	9.3	53.6	0.000	0.0	53.6	9.3	53.6
2.0	1.00	1.41	12.7	39.8	0.000	0.0	39.8	12.7	39.8
1.0	0.50	0.71	6.7	40.5	0.000	0.0	40.5	6.7	40.5
0.5	0.00	0.01	3.3	28.2	0.000	0.0	28.2	3.3	28.2
Total			100.0	56.6		53.5	65.0	46.5	46.9

Table A4.14: Screening simulation at a d_{50} of 8mm and Ep of 0.250

Size Fraction (mm)		Mean	Input Data		Simulated	Screen Oversize		Screen Undersize	
Upper	Lower	Size (mm)	Yield	Ash	Partition Number	Yield (%)	Ash (%)	Yield (%)	Ash (%)
20.0	16.00	17.89	26.3	69.8	1.000	26.3	69.8	0.0	69.8
16.0	8.00	11.31	27.3	60.3	1.000	27.3	60.3	0.0	60.3
8.0	4.00	5.66	14.4	56.1	0.000	0.0	56.1	14.4	56.1
4.0	2.00	2.83	9.3	53.6	0.000	0.0	53.6	9.3	53.6
2.0	1.00	1.41	12.7	39.8	0.000	0.0	39.8	12.7	39.8
1.0	0.50	0.71	6.7	40.5	0.000	0.0	40.5	6.7	40.5
0.5	0.00	0.01	3.3	28.2	0.000	0.0	28.2	3.3	28.2
Total			100.0	56.6		53.5	65.0	46.5	46.9

A4.2.3 Mutale Plant Discard Raw Washability Data by Size Fraction

Each size class was also subjected to a washability analysis, to determine the effect of destoning by dense medium separation, as detailed in Table A4.15 to Table A4.20 below:

Table A4.15: Washability data of the +16mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
+ 16 mm	Float 1.30	0.0					
	Float 1.35	0.0					
	Float 1.40	0.0					
	Float 1.45	0.1	0.7	19.4	19.5	0.1	19.5
	Float 1.50	0.8	0.7	24.0	24.2	0.9	23.8
	Float 1.55	1.2	0.7	27.9	28.1	2.1	26.3
	Float 1.60	1.4	0.8	32.1	32.4	3.5	28.7
	Float 1.65	2.0	0.7	35.3	35.5	5.5	31.2
	Float 1.70	2.0	0.8	38.8	39.1	7.5	33.3
	Float 1.75	2.1	0.8	42.6	42.9	9.5	35.4
	Float 1.80	1.7	0.6	47.2	47.5	11.3	37.3
	Float 1.85	1.9	0.6	48.9	49.2	13.2	39.0
	Float 1.90	2.0	0.9	51.6	52.0	15.2	40.7
	Sink 1.90	84.8	0.7	81.4	82.0	-	-
	Total	100.0	0.7	75.2	75.7	100.0	75.7

Table A4.16: Washability data of the -16+8mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 16 + 8 mm	Float 1.30	0.0					
	Float 1.35	0.0					
	Float 1.40	0.1	0.7	14.1	14.2	0.1	14.2
	Float 1.45	0.7	0.6	18.8	18.9	0.8	18.2
	Float 1.50	3.8	0.7	23.3	23.4	4.6	22.5
	Float 1.55	4.8	0.7	27.6	27.8	9.4	25.2
	Float 1.60	4.8	0.7	31.6	31.8	14.2	27.4
	Float 1.65	4.6	0.7	35.7	35.9	18.8	29.5
	Float 1.70	4.2	0.6	39.7	40.0	23.0	31.4
	Float 1.75	3.7	0.7	43.1	43.4	26.7	33.1
	Float 1.80	3.8	0.9	46.2	46.6	30.5	34.8
	Float 1.85	3.8	0.7	50.3	50.7	34.2	36.5
	Float 1.90	2.9	0.8	53.9	54.3	37.2	37.9
	Sink 1.90	62.8	0.8	77.0	77.6	-	-
	Total	100.0	0.8	62.4	62.9	100.0	62.9

Table A4.17: Washability data of the -8+4mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 8 + 4 mm	Float 1.30	0.0					
	Float 1.35	0.1	0.7	10.8	10.8	0.1	10.8
	Float 1.40	0.3	0.7	14.2	14.3	0.4	13.2
	Float 1.45	1.1	0.6	18.4	18.5	1.6	17.0
	Float 1.50	5.2	0.6	23.3	23.5	6.8	22.0
	Float 1.55	6.4	0.7	27.6	27.8	13.2	24.8
	Float 1.60	6.6	0.8	32.1	32.4	19.7	27.3
	Float 1.65	5.6	0.7	36.1	36.3	25.3	29.3
	Float 1.70	5.2	0.7	39.1	39.4	30.5	31.0
	Float 1.75	3.4	0.7	42.4	42.7	33.9	32.2
	Float 1.80	4.5	0.7	45.9	46.2	38.3	33.8
	Float 1.85	4.0	0.7	50.2	50.5	42.3	35.4
	Float 1.90	2.9	0.7	53.4	53.8	45.3	36.6
	Sink 1.90	54.7	1.0	75.9	76.7	-	-
	Total	100.0	0.8	58.0	58.5	100.0	58.5

Table A4.18: Washability data of the -4+2mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 4 + 2 mm	Float 1.30	2.4	0.6	5.6	5.6	2.4	5.6
	Float 1.35	1.9	0.6	10.1	10.1	4.2	7.6
	Float 1.40	1.9	0.7	13.7	13.8	6.2	9.5
	Float 1.45	2.8	0.7	18.1	18.3	9.0	12.3
	Float 1.50	5.1	0.5	23.5	23.6	14.1	16.4
	Float 1.55	5.6	0.8	27.4	27.6	19.7	19.6
	Float 1.60	6.3	0.7	31.6	31.8	25.9	22.5
	Float 1.65	5.2	0.8	35.6	35.9	31.2	24.8
	Float 1.70	5.0	0.9	39.2	39.6	36.2	26.8
	Float 1.75	2.6	0.8	42.4	42.8	38.7	27.9
	Float 1.80	4.6	0.9	46.0	46.4	43.4	29.9
	Float 1.85	4.5	0.8	50.7	51.1	47.9	31.9
	Float 1.90	2.4	0.8	54.7	55.2	50.3	33.0
	Sink 1.90	49.7	0.9	75.9	76.6	-	-
	Total	100.0	0.8	54.2	54.7	100.0	54.7

Table A4.19: Washability data of the -2+1mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 2 + 1 mm	Float 1.30	13.3	0.6	5.0	5.0	13.3	5.0
	Float 1.35	8.1	0.6	8.7	8.8	21.4	6.4
	Float 1.40	6.2	0.7	13.0	13.1	27.6	7.9
	Float 1.45	9.5	0.6	17.7	17.9	37.1	10.5
	Float 1.50	7.4	0.8	23.5	23.7	44.6	12.7
	Float 1.55	3.8	0.8	28.2	28.4	48.3	13.9
	Float 1.60	3.7	0.9	31.8	32.1	52.0	15.2
	Float 1.65	3.1	0.7	35.7	36.0	55.1	16.4
	Float 1.70	3.7	0.7	40.4	40.7	58.8	17.9
	Float 1.75	1.0	0.7	40.8	41.1	59.8	18.3
	Float 1.80	3.3	0.8	46.0	46.4	63.1	19.7
	Float 1.85	4.0	0.7	53.6	54.0	67.1	21.8
	Float 1.90	1.0	0.8	58.4	58.9	68.1	22.3
	Sink 1.90	31.9	0.6	74.4	74.9	-	-
	Total	100.0	0.7	38.8	39.1	100.0	39.1

Table A4.20: Washability data of the -1+0.5mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 1 + 0.5 mm	Float 1.30	16.9	0.8	6.7	6.8	16.9	6.8
	Float 1.35	5.6	0.7	10.0	10.1	22.5	7.6
	Float 1.40	6.3	0.7	14.3	14.4	28.8	9.1
	Float 1.45	7.5	0.8	18.4	18.6	36.3	11.1
	Float 1.50	8.4	0.7	23.3	23.5	44.7	13.4
	Float 1.55	3.8	0.8	28.3	28.5	48.5	14.6
	Float 1.60	5.2	0.8	31.7	32.0	53.7	16.3
	Float 1.65	2.7	0.9	36.5	36.8	56.4	17.3
	Float 1.70	3.7	0.9	41.3	41.6	60.2	18.8
	Float 1.75	1.3	0.8	43.7	44.1	61.4	19.3
	Float 1.80	3.8	0.8	49.8	50.2	65.3	21.1
	Float 1.85	6.0	0.7	60.9	61.4	71.3	24.5
	Float 1.90	1.3	0.8	64.0	64.5	72.5	25.2
	Sink 1.90	27.5	0.8	75.3	75.9	-	-
	Total	100.0	0.8	38.8	39.1	100.0	39.1

A4.2.4 Mutale Plant Waste Calculated Washability Data from Washability by Size Fraction Data

The washability data from above was then reconstituted to form a washability of the entire sample and also one for the -8+0.5mm size fraction. The latter was performed to allow for a circuit simulation utilizing both screening and dense medium separation as a method of destoning. The results are presented in Table A4.21 and Table A4.22.

Table A4.21: Reconstituted washability data of the whole discard (excluding the -0.5mm material)

Size Fraction	RD	Fractional		Cumulative	
		Yield (%)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
Total Sample	Float 1.30	5.4	6.0	5.4	6.0
	Float 1.35	2.6	9.4	8.0	7.1
	Float 1.40	2.5	13.8	10.5	8.7
	Float 1.45	3.6	18.2	14.1	11.1
	Float 1.50	5.1	23.6	19.3	14.4
	Float 1.55	4.2	28.0	23.5	16.9
	Float 1.60	4.7	32.0	28.2	19.4
	Float 1.65	3.9	36.1	32.0	21.4
	Float 1.70	4.0	40.1	36.0	23.5
	Float 1.75	2.3	42.9	38.3	24.6
	Float 1.80	3.6	47.2	42.0	26.6
	Float 1.85	4.0	53.8	46.0	29.0
	Float 1.90	2.1	55.4	48.1	30.1
	Sink 1.90	51.9	78.1	-	-
	Total	100.0	55.0	100.0	55.0

Table A4.22: Reconstituted washability data of the -8+0.5mm size fraction

Size Fraction	RD	Fractional		Cumulative	
		Yield (%)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
-8+0.5mm	Float 1.30	5.3	5.6	5.3	5.6
	Float 1.35	3.0	9.2	8.4	6.9
	Float 1.40	2.7	13.5	11.1	8.5
	Float 1.45	4.2	18.1	15.3	11.1
	Float 1.50	6.0	23.6	21.3	14.7
	Float 1.55	5.3	27.9	26.6	17.3
	Float 1.60	5.7	32.1	32.2	19.9
	Float 1.65	4.6	36.2	36.9	21.9
	Float 1.70	4.6	39.8	41.5	23.9
	Float 1.75	2.4	42.6	43.9	25.0
	Float 1.80	4.2	46.6	48.1	26.8
	Float 1.85	4.3	52.7	52.4	29.0
	Float 1.90	2.2	55.3	54.5	30.0
	Sink 1.90	45.5	76.3	-	-
	Total	100.0	51.1	100.0	51.1

A4.2.5 Mutale Plant Waste Screening and Dense Medium Separation Simulation

Using the reconstituted washability for the $-8+0.5\text{mm}$ size fraction, two simulations were performed, the first to produce a standard Tshikondeni coking coal product of 14% ash, the second was for the production of an alternative product suitable for supply to Eskom at an ash content as close as possible to a typical ash specification level of 30%. The screen was assumed to cut at 8mm with an Ep of 0.250 and the dense medium separation cyclone Ep was set at 0.025, which is within industry norms.

The simulations are illustrated in Figure A4.5 and Figure A4.6.

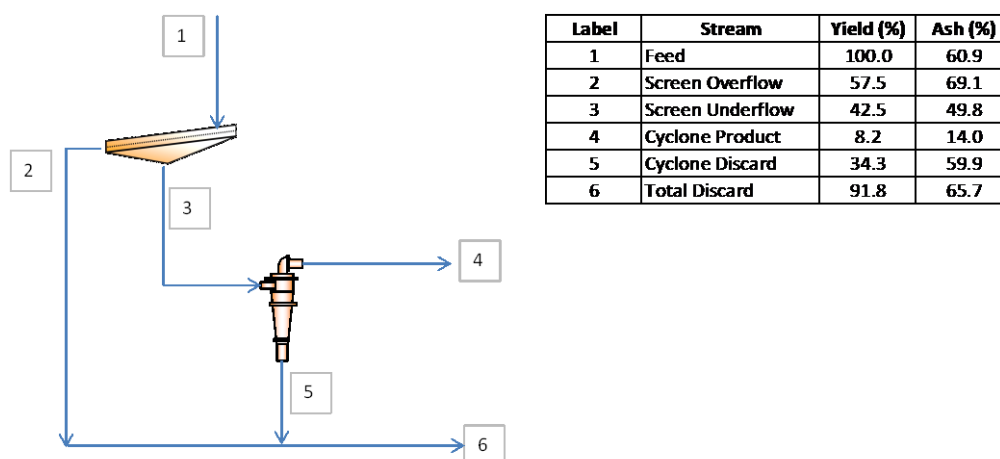


Figure A4.5: Particle size distribution of the sample as produced by the plant

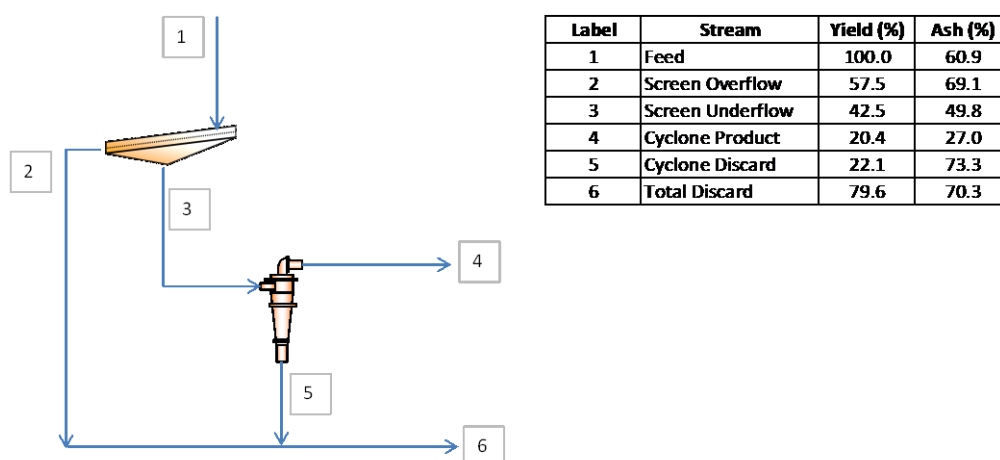


Figure A4.6: Particle size distribution of the sample as produced by the plant

A4.3 Vhukati Shaft

A4.3.1 Particle Size Distribution

The particle size distribution of the Vhukati Shaft discard sample was determined prior to any further processing and is presented in Table A4.146 and Figure A4.7.

Table A4.23: Particle size distribution of the sample as produced by the plant

Size (mm)	Mass (g)	Mass (%)	Passing (%)
20			100.00
16	33140	32.65	67.35
8	31560	31.10	36.25
4	14018	13.81	22.44
2	9322	9.18	13.25
1	8489	8.36	4.89
0.5	3008	2.96	1.93
0	1955	1.93	0
Total	101492	100.00	

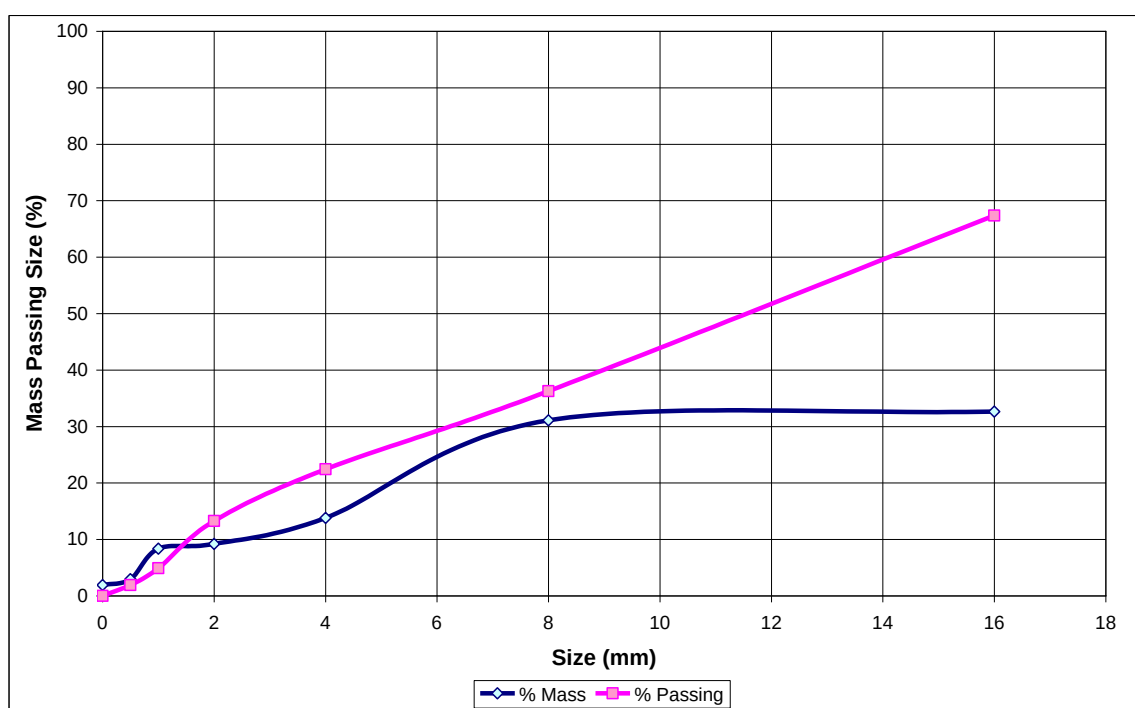


Figure A4.7: Particle size distribution of the sample as produced by the plant

A4.3.2 *Vhukati Shaft Screening Separation Simulations*

Having established the particle size distribution, each size fraction was analyzed for ash content and it was evident that 8mm would be a potentially suitable size to remove coarse high ash material and allow for destoning to take place. Two simulations were therefore performed using a d_{50} of 8mm and Ep values of 0.200 and 0.250 respectively. These are summarized in Table A4.24 and Table A4.25.

Table A4.24: Screening simulation at a d_{50} of 8mm and Ep of 0.200

Size Fraction (mm)		Mean	Input Data		Simulated	Screen Oversize		Screen Undersize	
Upper	Lower	Size (mm)	Yield	Ash	Partition Number	Yield (%)	Ash (%)	Yield (%)	Ash (%)
20.0	16.00	17.89	32.7	71.5	1.000	32.7	71.5	0.0	71.5
16.0	8.00	11.31	31.1	61.8	1.000	31.1	61.8	0.0	61.8
8.0	4.00	5.66	13.8	58.5	0.000	0.0	58.5	13.8	58.5
4.0	2.00	2.83	9.2	54.8	0.000	0.0	54.8	9.2	54.8
2.0	1.00	1.41	8.4	38.9	0.000	0.0	38.9	8.4	38.9
1.0	0.50	0.71	3.0	40.0	0.000	0.0	40.0	3.0	40.0
0.5	0.00	0.01	1.9	27.6	0.000	0.0	27.6	1.9	27.6
Total			100.0	60.7		63.7	66.8	36.3	49.9

Table A4.25: Screening simulation at a d_{50} of 8mm and Ep of 0.250

Size Fraction (mm)		Mean	Input Data		Simulated	Screen Oversize		Screen Undersize	
Upper	Lower	Size (mm)	Yield	Ash	Partition Number	Yield (%)	Ash (%)	Yield (%)	Ash (%)
20.0	16.00	17.89	32.7	71.5	1.000	32.7	71.5	0.0	71.5
16.0	8.00	11.31	31.1	61.8	1.000	31.1	61.8	0.0	61.8
8.0	4.00	5.66	13.8	58.5	0.000	0.0	58.5	13.8	58.5
4.0	2.00	2.83	9.2	54.8	0.000	0.0	54.8	9.2	54.8
2.0	1.00	1.41	8.4	38.9	0.000	0.0	38.9	8.4	38.9
1.0	0.50	0.71	3.0	40.0	0.000	0.0	40.0	3.0	40.0
0.5	0.00	0.01	1.9	27.6	0.000	0.0	27.6	1.9	27.6
Total			100.0	60.7		63.7	66.8	36.3	49.9

A4.3.3 Vhukati Shaft Raw Washability Data by Size Fraction

Each size class was also subjected to a washability analysis, to determine the effect of destoning by dense medium separation, as detailed in Table A4.26 to Table A4.31 below

Table A4.26: Washability data of the +16mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
+ 16 mm	Float 1.30	0.0					
	Float 1.35	0.03	0.7	17.0	17.1	0.03	17.1
	Float 1.40	0.0				0.03	17.1
	Float 1.45	0.0				0.03	17.1
	Float 1.50	0.2	0.7	24.7	24.9	0.2	23.8
	Float 1.55	1.4	0.6	28.6	28.8	1.6	28.1
	Float 1.60	0.9	0.5	32.1	32.3	2.5	29.6
	Float 1.65	2.2	0.6	36.3	36.5	4.7	32.8
	Float 1.70	2.1	0.7	39.9	40.2	6.8	35.1
	Float 1.75	3.0	0.7	42.0	42.2	9.8	37.3
	Float 1.80	3.2	0.6	47.6	47.9	13.0	39.9
	Float 1.85	3.0	0.7	51.5	51.9	16.0	42.1
	Float 1.90	2.4	0.6	54.5	54.8	18.4	43.8
	Sink 1.90	81.6	0.9	77.1	77.8	-	-
	Total	100.0	0.8	70.9	71.5	100.0	71.5

Table A4.27: Washability data of the -16+8mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 16 + 8 mm	Float 1.30	0.0					
	Float 1.35	0.0					
	Float 1.40	0.1	0.7	14.3	14.4	0.1	14.4
	Float 1.45	0.2	0.7	18.3	18.5	0.3	17.3
	Float 1.50	1.7	0.5	24.1	24.3	1.9	23.3
	Float 1.55	4.4	0.5	28.0	28.1	6.3	26.6
	Float 1.60	2.9	0.6	32.0	32.2	9.2	28.4
	Float 1.65	5.7	0.6	36.5	36.7	15.0	31.6
	Float 1.70	6.0	0.7	40.2	40.5	21.0	34.2
	Float 1.75	5.5	0.7	43.3	43.7	26.5	36.1
	Float 1.80	4.6	0.8	48.0	48.3	31.1	37.9
	Float 1.85	4.6	0.7	51.4	51.8	35.7	39.7
	Float 1.90	4.3	0.7	54.9	55.3	40.0	41.4
	Sink 1.90	60.0	0.8	74.8	75.4	-	-
	Total	100.0	0.7	61.3	61.8	100.0	61.8

Table A4.28: Washability data of the -8+4mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 8 + 4 mm	Float 1.30	0.0					
	Float 1.35	0.1	0.6	11.5	11.6	0.1	11.6
	Float 1.40	0.3	0.7	14.2	14.3	0.4	13.5
	Float 1.45	0.5	0.6	18.8	18.9	0.9	16.6
	Float 1.50	2.1	0.6	23.9	24.0	3.0	21.8
	Float 1.55	5.3	0.6	28.4	28.6	8.3	26.1
	Float 1.60	3.8	0.7	32.5	32.7	12.1	28.2
	Float 1.65	6.6	0.7	36.9	37.1	18.7	31.3
	Float 1.70	6.1	0.7	40.0	40.3	24.8	33.5
	Float 1.75	5.8	0.7	43.6	43.9	30.5	35.5
	Float 1.80	5.8	0.7	47.2	47.5	36.3	37.4
	Float 1.85	5.0	0.7	50.6	51.0	41.3	39.1
	Float 1.90	4.6	0.8	54.9	55.4	45.9	40.7
	Sink 1.90	54.1	0.8	73.1	73.7	-	-
	Total	100.0	0.7	58.1	58.5	100.0	58.5

Table A4.29: Washability data of the -4+2mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 4 + 2 mm	Float 1.30	2.1	0.7	5.8	5.9	2.1	5.9
	Float 1.35	1.5	0.7	8.5	8.6	3.6	7.0
	Float 1.40	2.2	0.6	13.8	13.9	5.8	9.6
	Float 1.45	2.2	0.6	18.0	18.1	8.0	11.9
	Float 1.50	3.9	0.7	23.4	23.6	11.9	15.7
	Float 1.55	4.5	0.6	28.6	28.7	16.4	19.3
	Float 1.60	3.0	0.8	31.9	32.1	19.4	21.3
	Float 1.65	6.0	0.6	36.5	36.7	25.4	24.9
	Float 1.70	5.9	0.8	40.3	40.6	31.3	27.9
	Float 1.75	4.8	0.7	43.4	43.6	36.1	30.0
	Float 1.80	5.5	0.7	47.5	47.9	41.6	32.3
	Float 1.85	3.7	0.8	51.1	51.5	45.2	33.9
	Float 1.90	4.4	0.8	54.6	55.0	49.6	35.8
	Sink 1.90	50.4	0.8	73.0	73.6	-	-
	Total	100.0	0.7	54.4	54.8	100.0	54.8

Table A4.30: Washability data of the -2+1mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 2 + 1 mm	Float 1.30	13.3	0.7	5.3	5.3	13.3	5.3
	Float 1.35	5.6	0.7	9.2	9.3	18.9	6.5
	Float 1.40	8.7	1.1	13.8	13.9	27.6	8.8
	Float 1.45	7.8	1.0	18.0	18.1	35.4	10.9
	Float 1.50	7.0	0.8	23.0	23.2	42.4	12.9
	Float 1.55	5.7	0.8	28.4	28.7	48.2	14.8
	Float 1.60	2.6	0.9	31.6	31.8	50.7	15.7
	Float 1.65	4.2	0.8	36.4	36.7	55.0	17.3
	Float 1.70	2.8	0.7	40.8	41.1	57.8	18.4
	Float 1.75	2.5	0.7	43.5	43.8	60.3	19.5
	Float 1.80	3.6	0.8	48.8	49.2	63.9	21.2
	Float 1.85	1.5	0.7	51.6	52.0	65.4	21.9
	Float 1.90	3.1	0.8	56.5	57.0	68.6	23.5
	Sink 1.90	31.4	0.8	71.9	72.5	-	-
	Total	100.0	0.8	38.6	38.9	100.0	38.9

Table A4.31: Washability data of the -1+0.5mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 1 + 0.5 mm	Float 1.30	11.6	0.8	6.4	6.5	11.6	6.5
	Float 1.35	4.9	0.7	10.1	10.2	16.6	7.6
	Float 1.40	7.1	0.8	15.0	15.2	23.7	9.9
	Float 1.45	8.4	0.9	19.2	19.4	32.1	12.4
	Float 1.50	9.2	0.8	23.7	23.9	41.2	14.9
	Float 1.55	6.9	1.0	30.5	30.9	48.1	17.2
	Float 1.60	3.8	0.8	31.9	32.2	52.0	18.3
	Float 1.65	3.5	0.7	37.2	37.5	55.5	19.5
	Float 1.70	3.9	1.2	42.6	43.1	59.4	21.1
	Float 1.75	1.5	0.7	44.3	44.7	60.9	21.6
	Float 1.80	5.9	0.8	52.6	53.0	66.7	24.4
	Float 1.85	1.0	0.9	54.7	55.2	67.7	24.9
	Float 1.90	4.3	0.9	62.0	62.6	72.1	27.1
	Sink 1.90	27.9	0.8	72.8	73.4	-	-
	Total	100.0	0.8	39.7	40.0	100.0	40.0

A4.3.4 Vhukati Shaft Calculated Washability Data from Washability by Size Fraction Data

The washability data from above was then reconstituted to form a washability of the entire sample and also one for the -8+0.5mm size fraction. The latter was performed to allow for a circuit simulation utilizing both screening and dense medium separation as a method of destoning. The results are presented in Table A4.32. and Table A4.33.

Table A4.32: Reconstituted washability data of the whole discard (excluding the -0.5mm material)

Size Fraction	RD	Fractional		Cumulative	
		Yield (%)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
Total Sample	Float 1.30	4.5	5.9	4.5	5.9
	Float 1.35	2.0	9.6	6.5	7.0
	Float 1.40	3.1	14.4	9.6	9.4
	Float 1.45	3.2	18.7	12.8	11.7
	Float 1.50	4.0	23.7	16.8	14.6
	Float 1.55	4.7	29.1	21.5	17.7
	Float 1.60	2.8	32.2	24.3	19.4
	Float 1.65	4.7	36.9	29.0	22.3
	Float 1.70	4.5	40.9	33.5	24.7
	Float 1.75	3.9	43.6	37.3	26.7
	Float 1.80	4.7	49.1	42.1	29.2
	Float 1.85	3.1	51.7	45.2	30.8
	Float 1.90	3.9	56.8	49.1	32.8
	Sink 1.90	50.9	74.9	-	-
	Total	100.0	54.3	100.0	54.3

Table A4.33: Reconstituted washability data of the -8+0.5mm size fraction

Size Fraction	RD	Fractional		Cumulative	
		Yield (%)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
-8+0.5mm	Float 1.30	4.8	5.6	4.8	5.6
	Float 1.35	2.2	9.4	7.0	6.8
	Float 1.40	3.4	14.1	10.4	9.2
	Float 1.45	3.4	18.4	13.8	11.5
	Float 1.50	4.4	23.6	18.2	14.4
	Float 1.55	5.3	28.9	23.5	17.7
	Float 1.60	3.3	32.3	26.8	19.5
	Float 1.65	5.6	37.0	32.4	22.5
	Float 1.70	5.1	40.7	37.4	25.0
	Float 1.75	4.4	43.8	41.8	26.9
	Float 1.80	5.2	48.4	47.0	29.3
	Float 1.85	3.5	51.4	50.4	30.8
	Float 1.90	4.2	56.2	54.6	32.7
	Sink 1.90	45.4	73.4	-	-
	Total	100.0	51.2	100.0	51.2

A4.3.5 Vhukati Shaft Screening and Dense Medium Separation Simulation

Using the reconstituted washability for the $-8+0.5\text{mm}$ size fraction, two simulations were performed, the first to produce a standard Tshikondeni coking coal product of 14% ash, the second was for the production of an alternative product suitable for supply to Eskom at an ash content as close as possible to a typical ash specification level of 30%. The screen was assumed to cut at 8mm with an Ep of 0.250 and the dense medium separation cyclone Ep was set at 0.025, which is within industry norms.

The simulations are illustrated in Figure A4.8 and Figure A4.9.

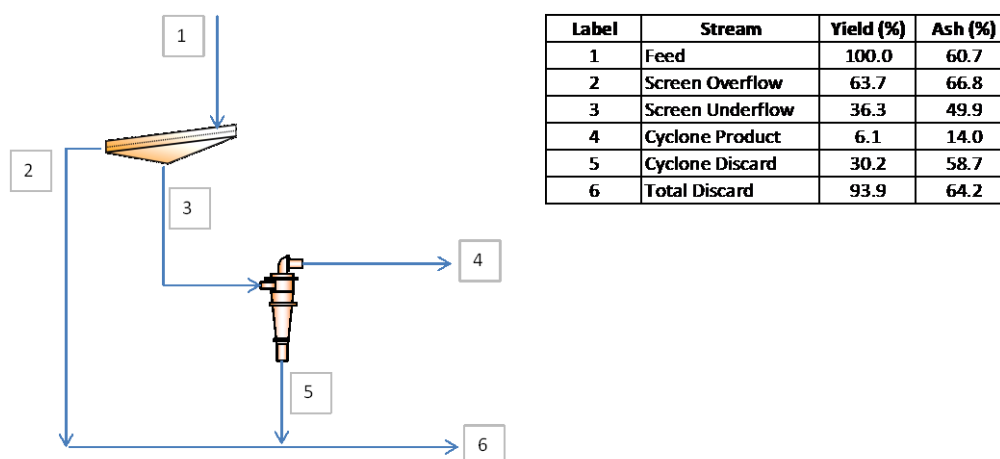


Figure A4.8: Particle size distribution of the sample as produced by the plant

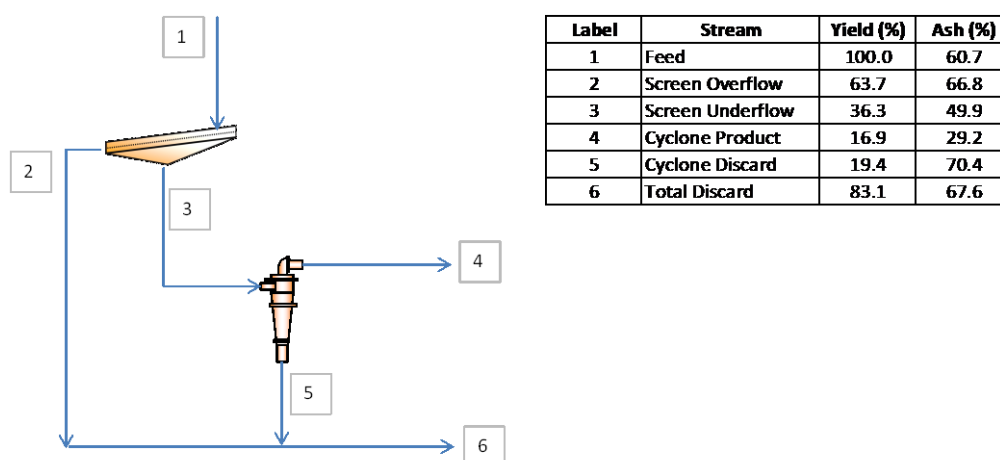


Figure A4.9: Particle size distribution of the sample as produced by the plant

A4.4 Vhukati Plant Waste

A4.4.1 Particle Size Distribution

The particle size distribution of the Vhukati Plant Waste discard sample was determined prior to any further processing and is presented in Table A4.34. and Figure A4.10.

Table A4.34: Particle size distribution of the sample as produced by the plant

Size (mm)	Mass (g)	Mass (%)	Passing (%)
20			100.00
16	23580	23.46	76.54
8	32360	32.20	44.34
4	15173	15.10	29.25
2	10520	10.47	18.78
1	13565	13.50	5.28
0.5	3327	3.31	1.97
0	1983	1.97	0
Total	100508	100.00	

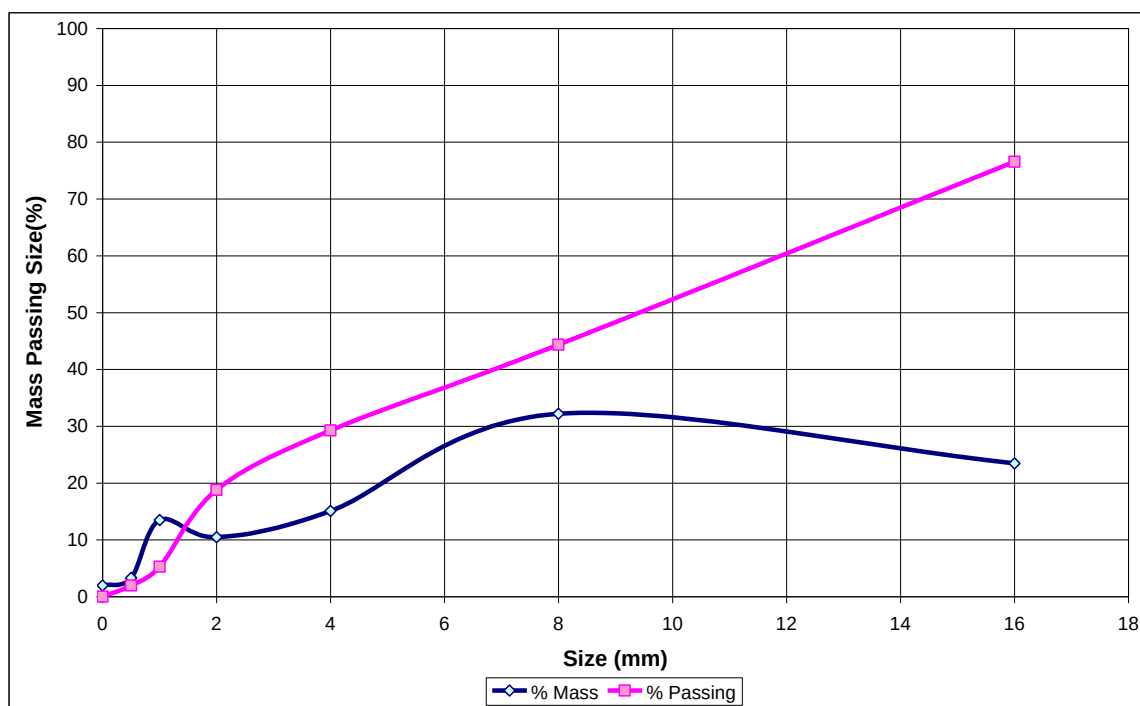


Figure A4.10: Particle size distribution of the sample as produced by the plant

A4.4.2 Vhukati Plant Waste Screening Separation Simulations

Having established the particle size distribution, each size fraction was analyzed for ash content and it was evident that 8mm would be a potentially suitable size to remove coarse high ash material and allow for destoning to take place. Two simulations were therefore performed using a d_{50} of 8mm and Ep values of 0.200 and 0.250 respectively. These are summarized in Table A4.35 and Table A4.36.

Table A4.35: Screening simulation at a d_{50} of 8mm and Ep of 0.200

Size Fraction (mm)		Mean	Input Data		Simulated	Screen Oversize		Screen Undersize	
Upper	Lower	Size (mm)	Yield	Ash	Partition Number	Yield (%)	Ash (%)	Yield (%)	Ash (%)
20.0	16.00	17.89	23.5	69.9	1.000	23.5	69.9	0.0	69.9
16.0	8.00	11.31	32.2	61.8	1.000	32.2	61.8	0.0	61.8
8.0	4.00	5.66	15.1	58.2	0.000	0.0	58.2	15.1	58.2
4.0	2.00	2.83	10.5	52.8	0.000	0.0	52.8	10.5	52.8
2.0	1.00	1.41	13.5	36.7	0.000	0.0	36.7	13.5	36.7
1.0	0.50	0.71	3.3	41.6	0.000	0.0	41.6	3.3	41.6
0.5	0.00	0.01	2.0	26.5	0.000	0.0	26.5	2.0	26.5
Total			100.0	57.5		55.7	65.2	44.3	47.7

Table A4.36: Screening simulation at a d_{50} of 8mm and Ep of 0.250

Size Fraction (mm)		Mean	Input Data		Simulated	Screen Oversize		Screen Undersize	
Upper	Lower	Size (mm)	Yield	Ash	Partition Number	Yield (%)	Ash (%)	Yield (%)	Ash (%)
20.0	16.00	17.89	23.5	69.9	1.000	23.5	69.9	0.0	69.9
16.0	8.00	11.31	32.2	61.8	1.000	32.2	61.8	0.0	61.8
8.0	4.00	5.66	15.1	58.2	0.000	0.0	58.2	15.1	58.2
4.0	2.00	2.83	10.5	52.8	0.000	0.0	52.8	10.5	52.8
2.0	1.00	1.41	13.5	36.7	0.000	0.0	36.7	13.5	36.7
1.0	0.50	0.71	3.3	41.6	0.000	0.0	41.6	3.3	41.6
0.5	0.00	0.01	2.0	26.5	0.000	0.0	26.5	2.0	26.5
Total			100.0	57.5		55.7	65.2	44.3	47.7

A4.4.3 Vhukati Plant Waste Raw Washability Data by Size Fraction

Each size class was also subjected to a washability analysis, to determine the effect of destoning by dense medium separation, as detailed in Table A4.37 to Table A4.42 below:

Table A4.37: Washability data of the +16mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
+ 16 mm	Float 1.30	0.0					
	Float 1.35	0.0					
	Float 1.40	0.1	0.7	27.4	27.6	0.1	27.6
	Float 1.45	0.0				0.1	27.6
	Float 1.50	0.1	0.7	24.4	24.5	0.2	25.5
	Float 1.55	1.2	0.6	28.0	28.2	1.4	27.8
	Float 1.60	1.2	0.7	31.7	31.9	2.6	29.7
	Float 1.65	1.9	0.7	37.3	37.6	4.5	33.0
	Float 1.70	2.6	0.6	40.0	40.3	7.1	35.7
	Float 1.75	2.7	0.7	43.6	43.9	9.8	38.0
	Float 1.80	3.2	0.7	47.5	47.9	13.0	40.4
	Float 1.85	4.4	0.8	51.4	51.8	17.4	43.2
	Float 1.90	2.3	0.9	54.6	55.1	19.6	44.6
	Sink 1.90	80.4	0.8	75.5	76.1	-	-
	Total	100.0	0.7	69.4	69.9	100.0	69.9

Table A4.38: Washability data of the -16+8mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 16 + 8 mm	Float 1.30	0.0					
	Float 1.35	0.0					
	Float 1.40	0.1	0.7	13.2	13.3	0.1	13.3
	Float 1.45	0.1	0.8	16.8	16.9	0.2	15.3
	Float 1.50	0.7	0.7	23.5	23.7	0.9	21.8
	Float 1.55	2.9	0.8	27.5	27.7	3.7	26.3
	Float 1.60	4.5	0.7	31.8	32.0	8.2	29.4
	Float 1.65	5.9	0.7	36.6	36.8	14.1	32.5
	Float 1.70	5.7	0.7	39.9	40.2	19.7	34.7
	Float 1.75	5.3	0.7	43.6	43.9	25.0	36.7
	Float 1.80	5.1	0.7	47.4	47.7	30.1	38.5
	Float 1.85	5.5	0.7	51.1	51.4	35.7	40.5
	Float 1.90	4.1	0.7	54.2	54.7	39.8	42.0
	Sink 1.90	60.2	0.9	74.2	74.9	-	-
	Total	100.0	0.8	61.3	61.8	100.0	61.8

Table A4.39: Washability data of the -8+4mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 8 + 4 mm	Float 1.30	0.0					
	Float 1.35	0.1	0.7	10.1	10.2	0.1	10.2
	Float 1.40	0.3	0.6	13.6	13.7	0.4	12.6
	Float 1.45	0.4	0.6	17.8	17.9	0.8	15.2
	Float 1.50	1.6	0.5	23.4	23.5	2.4	20.7
	Float 1.55	4.3	0.6	28.2	28.4	6.8	25.6
	Float 1.60	6.3	0.6	31.9	32.1	13.0	28.8
	Float 1.65	6.0	0.7	36.2	36.5	19.1	31.2
	Float 1.70	6.6	0.7	39.3	39.5	25.7	33.4
	Float 1.75	5.8	0.6	43.2	43.5	31.5	35.2
	Float 1.80	5.7	0.7	47.3	47.7	37.2	37.1
	Float 1.85	5.8	0.7	51.2	51.6	42.9	39.1
	Float 1.90	3.9	0.8	54.6	55.1	46.9	40.4
	Sink 1.90	53.1	0.8	73.3	73.9	-	-
	Total	100.0	0.8	57.8	58.2	100.0	58.2

Table A4.40: Washability data of the -4+2mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 4 + 2 mm	Float 1.30	2.6	0.5	5.3	5.3	2.6	5.3
	Float 1.35	2.3	0.6	9.0	9.0	4.9	7.1
	Float 1.40	2.8	0.6	12.4	12.5	7.7	9.0
	Float 1.45	2.6	0.6	17.0	17.1	10.3	11.1
	Float 1.50	4.2	0.7	22.7	22.9	14.4	14.5
	Float 1.55	4.9	0.7	29.0	29.2	19.3	18.2
	Float 1.60	5.5	0.7	33.2	33.4	24.8	21.6
	Float 1.65	4.8	0.7	36.6	36.8	29.6	24.1
	Float 1.70	5.5	0.7	39.9	40.2	35.2	26.6
	Float 1.75	4.6	0.6	43.2	43.5	39.8	28.6
	Float 1.80	5.1	0.6	47.2	47.5	44.9	30.7
	Float 1.85	5.6	0.7	52.3	52.7	50.5	33.2
	Float 1.90	3.2	0.7	55.6	56.0	53.7	34.5
	Sink 1.90	46.3	0.8	73.4	74.0	-	-
	Total	100.0	0.7	52.4	52.8	100.0	52.8

Table A4.41: Washability data of the -2+1mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 2 + 1 mm	Float 1.30	10.0	0.6	4.5	4.5	10.0	4.5
	Float 1.35	12.5	0.6	9.7	9.8	22.5	7.5
	Float 1.40	5.5	0.6	13.2	13.3	28.1	8.6
	Float 1.45	9.5	0.6	17.6	17.7	37.6	10.9
	Float 1.50	9.2	1.0	22.7	23.0	46.8	13.3
	Float 1.55	5.4	0.8	28.7	28.9	52.2	14.9
	Float 1.60	3.5	0.6	32.6	32.8	55.7	16.0
	Float 1.65	2.3	0.6	35.4	35.6	58.1	16.8
	Float 1.70	3.7	0.9	39.8	40.2	61.8	18.2
	Float 1.75	2.1	0.8	43.0	43.3	63.9	19.0
	Float 1.80	2.9	0.9	48.0	48.5	66.8	20.3
	Float 1.85	3.4	0.9	53.8	54.3	70.2	22.0
	Float 1.90	1.9	0.9	56.2	56.8	72.1	22.9
	Sink 1.90	27.9	1.1	71.5	72.4	-	-
	Total	100.0	0.8	36.3	36.7	100.0	36.7

Table A4.42: Washability data of the -1+0.5mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 1 + 0.5 mm	Float 1.30	6.7	0.6	4.6	4.6	6.7	4.6
	Float 1.35	6.5	0.7	9.6	9.7	13.2	7.1
	Float 1.40	5.5	0.7	13.4	13.5	18.7	9.0
	Float 1.45	9.2	0.7	18.0	18.1	27.9	12.0
	Float 1.50	10.7	1.0	23.6	23.9	38.7	15.3
	Float 1.55	6.2	0.8	29.9	30.2	44.9	17.4
	Float 1.60	4.2	0.7	33.4	33.6	49.0	18.7
	Float 1.65	4.5	0.7	37.6	37.9	53.5	20.3
	Float 1.70	3.2	0.7	41.6	41.9	56.7	21.6
	Float 1.75	2.6	0.8	45.7	46.1	59.3	22.6
	Float 1.80	3.1	0.8	50.8	51.3	62.5	24.1
	Float 1.85	5.5	0.9	57.9	58.5	68.0	26.9
	Float 1.90	3.1	0.9	61.4	62.0	71.0	28.4
	Sink 1.90	29.0	1.0	73.3	74.0	-	-
	Total	100.0	0.8	41.2	41.6	100.0	41.6

A4.4.4 Vhukati Plant Waste Calculated Washability Data from Washability by Size Fraction Data

The washability data from above was then reconstituted to form a washability of the entire sample and also one for the -8+0.5mm size fraction. The latter was performed to allow for a circuit simulation utilizing both screening and dense medium separation as a method of destoning. The results are presented in Table A4.43 and Table A4.44.

Table A4.43: Reconstituted washability data of the whole discard (excluding the -0.5mm material)

Size Fraction	RD	Fractional		Cumulative	
		Yield (%)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
Total Sample	Float 1.30	3.2	4.7	3.2	4.7
	Float 1.35	3.6	9.7	6.8	7.3
	Float 1.40	2.4	13.3	9.2	8.9
	Float 1.45	3.6	17.8	12.8	11.4
	Float 1.50	4.4	23.4	17.2	14.5
	Float 1.55	4.1	29.0	21.4	17.3
	Float 1.60	4.2	32.7	25.6	19.8
	Float 1.65	4.2	36.9	29.8	22.2
	Float 1.70	4.6	40.3	34.4	24.6
	Float 1.75	3.9	43.9	38.2	26.6
	Float 1.80	4.2	48.2	42.4	28.7
	Float 1.85	5.0	53.3	47.4	31.3
	Float 1.90	3.1	56.5	50.5	32.9
	Sink 1.90	49.5	74.6	-	-
	Total	100.0	53.5	100.0	53.5

Table A4.44: Reconstituted washability data of the -8+0.5mm size fraction

Size Fraction	RD	Fractional		Cumulative	
		Yield (%)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
-8+0.5mm	Float 1.30	4.3	4.7	4.3	4.7
	Float 1.35	5.1	9.7	9.4	7.4
	Float 1.40	3.0	13.1	12.4	8.8
	Float 1.45	4.5	17.7	16.9	11.1
	Float 1.50	5.3	23.1	22.2	14.0
	Float 1.55	4.9	28.9	27.2	16.7
	Float 1.60	5.0	32.7	32.2	19.2
	Float 1.65	4.4	36.5	36.6	21.3
	Float 1.70	5.2	40.0	41.8	23.6
	Float 1.75	4.1	43.6	45.9	25.4
	Float 1.80	4.5	48.0	50.4	27.4
	Float 1.85	5.0	53.0	55.3	29.7
	Float 1.90	3.1	56.1	58.4	31.1
	Sink 1.90	41.6	73.6	-	-
	Total	100.0	48.8	100.0	48.8

A4.4.5 Vhukati Plant Waste Screening and Dense Medium Separation Simulation

Using the reconstituted washability for the -8+0.5mm size fraction, two simulations were performed, the first to produce a standard Tshikondeni coking coal product of 14% ash, the second was for the production of an alternative product suitable for supply to Eskom at an ash content as close as possible to a typical ash specification level of 30%. The screen was assumed to cut at 8mm with an Ep of 0.250 and the dense medium separation cyclone Ep was set at 0.025, which is within industry norms.

The simulations are illustrated in Figure A4.11 and Figure A4.12.

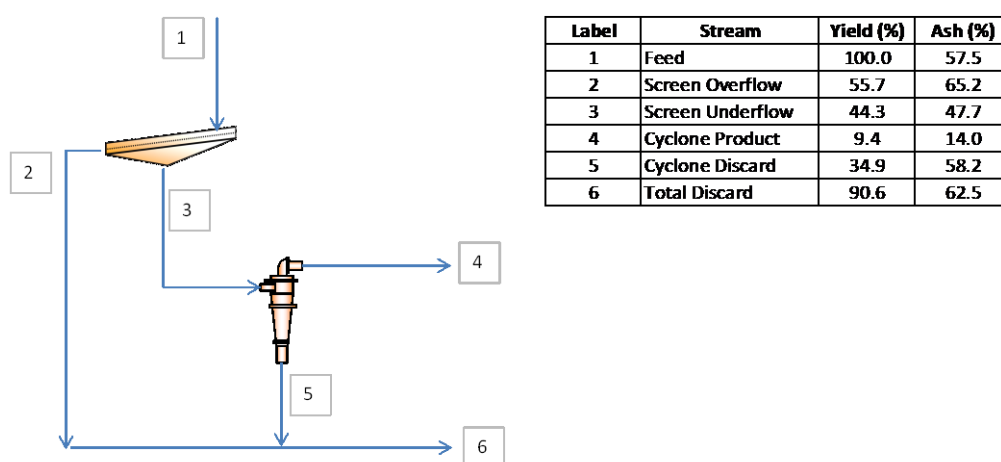


Figure A4.11: Particle size distribution of the sample as produced by the plant

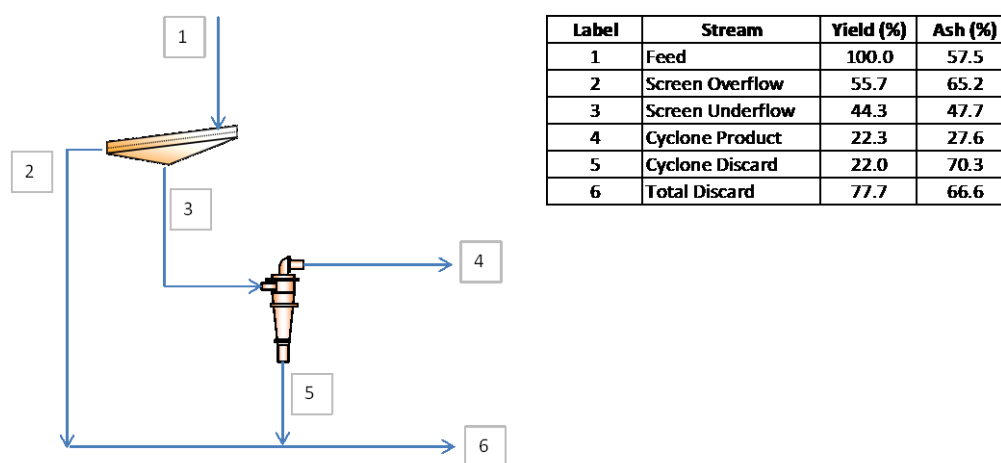


Figure A4.12: Particle size distribution of the sample as produced by the plant

A4.5 Goni Plant Waste

A4.5.1 Particle Size Distribution

The particle size distribution of the Goni Plant Waste discard sample was determined prior to any further processing and is presented in Table A4.45 and Figure A4.13.

Table A4.45: Particle size distribution of the sample as produced by the plant

Size (mm)	Mass (g)	Mass (%)	Passing (%)
20			100.00
16	23360	23.23	76.77
8	29780	29.61	47.16
4	15308	15.22	31.94
2	10374	10.32	21.62
1	16043	15.95	5.67
0.5	3546	3.53	2.15
0	2158	2.15	0
Total	100569	100.00	

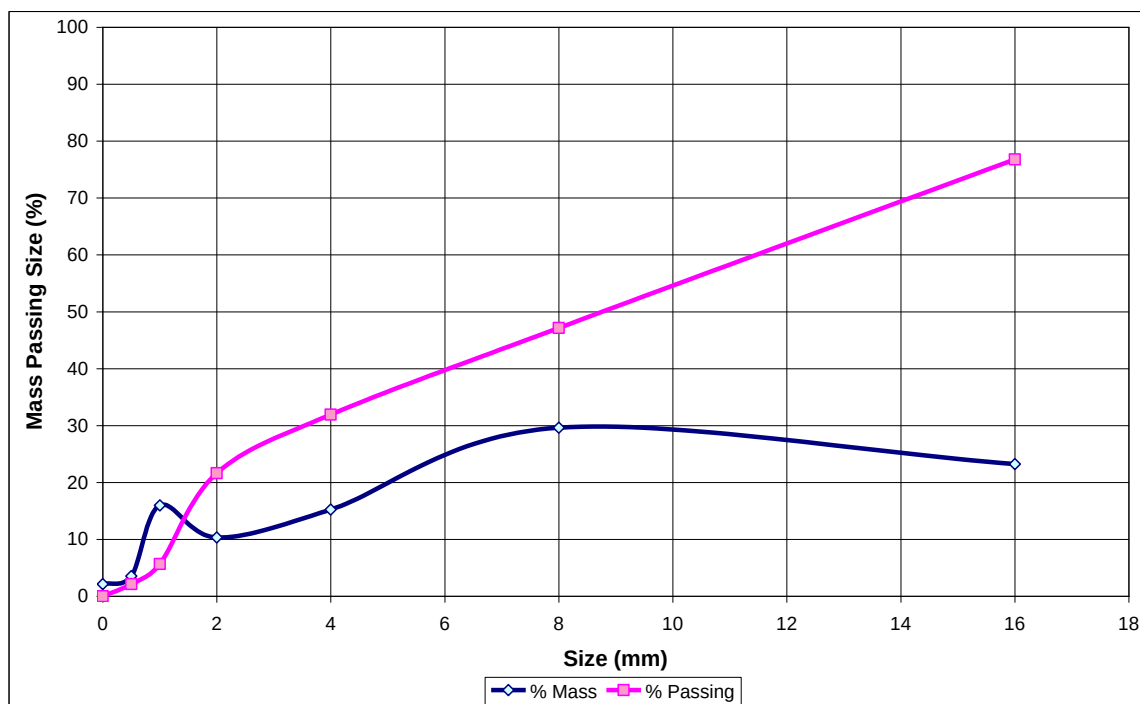


Figure A4.13: Particle size distribution of the sample as produced by the plant

A4.5.2 Goni Plant Waste Screening Separation Simulations

Having established the particle size distribution, each size fraction was analyzed for ash content and it was evident that 8mm would be a potentially suitable size to remove coarse high ash material and allow for destoning to take place. Two simulations were therefore performed using a d_{50} of 8mm and Ep values of 0.200 and 0.250 respectively. These are summarized in Table A4.46 and Table A4.47.

Table A4.46: Screening simulation at a d_{50} of 8mm and Ep of 0.200

Size Fraction (mm)		Mean	Input Data		Simulated	Screen Oversize		Screen Undersize	
Upper	Lower	Size (mm)	Yield	Ash	Partition Number	Yield (%)	Ash (%)	Yield (%)	Ash (%)
20.0	16.00	17.89	23.2	74.1	1.000	23.2	74.1	0.0	74.1
16.0	8.00	11.31	29.6	66.8	1.000	29.6	66.8	0.0	66.8
8.0	4.00	5.66	15.2	63.5	0.000	0.0	63.5	15.2	63.5
4.0	2.00	2.83	10.3	55.3	0.000	0.0	55.3	10.3	55.3
2.0	1.00	1.41	16.0	32.4	0.000	0.0	32.4	16.0	32.4
1.0	0.50	0.71	3.5	36.6	0.000	0.0	36.6	3.5	36.6
0.5	0.00	0.01	2.1	27.0	0.000	0.0	27.0	2.1	27.0
Total			100.0	59.4		52.8	70.0	47.1	47.5

Table A4.47: Screening simulation at a d_{50} of 8mm and Ep of 0.250

Size Fraction (mm)		Mean	Input Data		Simulated	Screen Oversize		Screen Undersize	
Upper	Lower	Size (mm)	Yield	Ash	Partition Number	Yield (%)	Ash (%)	Yield (%)	Ash (%)
20.0	16.00	17.89	23.2	74.1	1.000	23.2	74.1	0.0	74.1
16.0	8.00	11.31	29.6	66.8	1.000	29.6	66.8	0.0	66.8
8.0	4.00	5.66	15.2	63.5	0.000	0.0	63.5	15.2	63.5
4.0	2.00	2.83	10.3	55.3	0.000	0.0	55.3	10.3	55.3
2.0	1.00	1.41	16.0	32.4	0.000	0.0	32.4	16.0	32.4
1.0	0.50	0.71	3.5	36.6	0.000	0.0	36.6	3.5	36.6
0.5	0.00	0.01	2.1	27.0	0.000	0.0	27.0	2.1	27.0
Total			100.0	59.4		52.8	70.0	47.1	47.5

A4.5.3 Goni Plant Waste Raw Washability Data by Size Fraction

Each size class was also subjected to a washability analysis, to determine the effect of destoning by dense medium separation, as detailed in Table A4.49 to Table A4.54 below

Table A4.48: Washability data of the +16mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
+ 16 mm	Float 1.30	0.0					
	Float 1.35	0.1	0.7	15.3	15.4	0.1	15.4
	Float 1.40	0.0				0.1	15.4
	Float 1.45	0.1	0.6	20.7	20.8	0.1	18.1
	Float 1.50	0.0				0.1	18.1
	Float 1.55	0.4	0.6	29.0	29.2	0.6	26.7
	Float 1.60	0.7	0.7	33.3	33.5	1.3	30.6
	Float 1.65	0.7	0.7	35.9	36.1	2.0	32.5
	Float 1.70	1.5	0.8	41.6	41.9	3.4	36.6
	Float 1.75	2.3	0.8	44.4	44.8	5.8	39.9
	Float 1.80	2.1	0.7	48.1	48.4	7.9	42.2
	Float 1.85	3.1	0.7	53.1	53.5	11.0	45.4
	Float 1.90	2.6	0.7	55.7	56.1	13.6	47.4
	Sink 1.90	86.4	0.8	77.7	78.3	-	-
	Total	100.0	0.7	73.5	74.1	100.0	74.1

Table A4.49: Washability data of the -16+8mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 16 + 8 mm	Float 1.30	0.0					
	Float 1.35	0.2	0.6	11.1	11.1	0.2	11.1
	Float 1.40	0.2	0.6	14.7	14.8	0.4	13.1
	Float 1.45	0.2	0.7	18.5	18.6	0.6	14.9
	Float 1.50	0.2	0.6	23.2	23.4	0.7	17.0
	Float 1.55	1.3	0.7	29.0	29.2	2.1	24.7
	Float 1.60	2.1	0.7	33.2	33.4	4.2	29.1
	Float 1.65	2.2	0.6	36.6	36.8	6.4	31.8
	Float 1.70	3.7	0.8	40.4	40.7	10.0	35.0
	Float 1.75	3.7	0.7	44.2	44.5	13.8	37.6
	Float 1.80	4.5	0.8	48.9	49.3	18.2	40.5
	Float 1.85	6.2	0.8	52.5	52.9	24.5	43.6
	Float 1.90	3.4	0.8	55.4	55.9	27.9	45.1
	Sink 1.90	72.1	1.0	74.5	75.2	-	-
	Total	100.0	0.9	66.2	66.8	100.0	66.8

Table A4.50: Washability data of the -8+4mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 8 + 4 mm	Float 1.30	0.3	0.7	7.5	7.6	0.3	7.6
	Float 1.35	0.7	0.6	11.0	11.1	0.9	10.1
	Float 1.40	0.6	0.7	15.2	15.3	1.5	12.1
	Float 1.45	0.6	0.7	19.1	19.2	2.1	14.1
	Float 1.50	0.7	0.7	24.2	24.4	2.8	16.6
	Float 1.55	2.1	0.7	29.1	29.3	4.8	22.0
	Float 1.60	2.9	0.8	32.7	33.0	7.8	26.2
	Float 1.65	3.8	0.7	36.9	37.1	11.5	29.8
	Float 1.70	4.2	0.6	40.8	41.0	15.7	32.8
	Float 1.75	4.4	0.6	44.4	44.7	20.1	35.4
	Float 1.80	4.5	0.7	48.3	48.7	24.6	37.8
	Float 1.85	6.7	0.7	52.3	52.7	31.4	41.0
	Float 1.90	4.3	0.7	55.5	55.9	35.7	42.8
	Sink 1.90	64.3	1.0	74.2	74.9	-	-
	Total	100.0	0.9	62.9	63.5	100.0	63.5

Table A4.51: Washability data of the -4+2mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 4 + 2 mm	Float 1.30	4.4	0.7	5.5	5.6	4.4	5.6
	Float 1.35	4.4	0.8	10.0	10.1	8.7	7.8
	Float 1.40	3.3	0.7	15.4	15.5	12.1	9.9
	Float 1.45	2.8	0.6	19.0	19.1	14.9	11.7
	Float 1.50	2.3	0.7	24.0	24.2	17.2	13.4
	Float 1.55	2.6	0.7	29.3	29.5	19.8	15.5
	Float 1.60	3.2	0.7	33.0	33.2	23.0	17.9
	Float 1.65	3.5	0.7	37.3	37.6	26.5	20.5
	Float 1.70	3.5	0.6	40.6	40.8	30.0	22.9
	Float 1.75	4.0	0.6	45.0	45.3	33.9	25.5
	Float 1.80	3.4	0.7	48.1	48.4	37.3	27.6
	Float 1.85	4.9	0.8	52.2	52.6	42.2	30.5
	Float 1.90	5.0	0.7	56.1	56.4	47.3	33.2
	Sink 1.90	52.7	0.9	74.4	75.0	-	-
	Total	100.0	0.8	54.8	55.3	100.0	55.3

Table A4.52: Washability data of the -2+1mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 2 + 1 mm	Float 1.30	22.0	0.7	5.7	5.7	22.0	5.7
	Float 1.35	11.4	0.7	11.4	11.4	33.4	7.7
	Float 1.40	9.5	0.7	15.4	15.5	42.9	9.4
	Float 1.45	8.0	0.7	19.7	19.8	50.9	11.1
	Float 1.50	6.0	0.8	24.2	24.3	56.9	12.5
	Float 1.55	4.1	0.8	28.7	28.9	61.0	13.6
	Float 1.60	3.7	0.8	33.4	33.6	64.7	14.7
	Float 1.65	2.2	0.7	37.5	37.8	66.9	15.5
	Float 1.70	2.2	0.7	41.7	42.0	69.1	16.3
	Float 1.75	2.2	0.7	46.0	46.3	71.4	17.3
	Float 1.80	1.3	0.8	49.5	49.9	72.7	17.9
	Float 1.85	2.5	0.8	53.5	53.9	75.1	19.0
	Float 1.90	1.8	0.8	57.1	57.6	76.9	19.9
	Sink 1.90	23.1	1.0	73.1	73.8	-	-
	Total	100.0	0.8	32.1	32.4	100.0	32.4

Table A4.53: Washability data of the -1+0.5mm size fraction

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
- 1 + 0.5 mm	Float 1.30	16.7	0.7	7.2	7.3	16.7	7.3
	Float 1.35	7.3	0.7	12.8	12.9	24.0	9.0
	Float 1.40	8.7	0.6	16.6	16.7	32.8	11.0
	Float 1.45	6.5	0.7	19.8	19.9	39.3	12.5
	Float 1.50	9.4	0.7	24.4	24.6	48.8	14.8
	Float 1.55	4.6	0.7	29.3	29.4	53.4	16.1
	Float 1.60	6.1	0.8	33.6	33.9	59.5	17.9
	Float 1.65	4.0	0.8	39.3	39.6	63.5	19.3
	Float 1.70	2.6	0.8	41.7	42.0	66.1	20.2
	Float 1.75	3.9	0.8	49.1	49.4	70.0	21.8
	Float 1.80	1.0	0.9	52.9	53.4	71.0	22.3
	Float 1.85	2.5	0.9	57.4	57.9	73.5	23.5
	Float 1.90	3.7	0.9	63.7	64.3	77.2	25.5
	Sink 1.90	22.8	0.9	73.6	74.2	-	-
	Total	100.0	0.8	36.3	36.6	100.0	36.6

A4.5.4 Goni Plant Waste Calculated Washability Data from Washability by Size Fraction Data

The washability data from above was then reconstituted to form a washability of the entire sample and also one for the -8+0.5mm size fraction. The latter was performed to allow for a circuit simulation utilizing both screening and dense medium separation as a method of destoning. The results are presented in Table A4.54 and Table A4.55.

Table A4.54: Reconstituted washability data of the whole discard (excluding the -0.5mm material)

Size Fraction	RD	Fractional				Cumulative	
		Yield (%)	Moisture (%)	Ash (%) (AD)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
Total Sample	Float 1.30	7.2			6.3	7.2	6.3
	Float 1.35	4.0			11.6	11.2	8.2
	Float 1.40	3.7			15.9	15.0	10.1
	Float 1.45	3.0			19.7	18.0	11.8
	Float 1.50	3.1			24.4	21.1	13.6
	Float 1.55	2.5			29.3	23.6	15.3
	Float 1.60	3.1			33.5	26.7	17.4
	Float 1.65	2.7			37.8	29.5	19.3
	Float 1.70	2.9			41.3	32.4	21.3
	Float 1.75	3.4			45.9	35.8	23.7
	Float 1.80	2.8			49.1	38.6	25.5
	Float 1.85	4.3			53.4	42.9	28.3
	Float 1.90	3.5			57.7	46.4	30.5
	Sink 1.90	53.6			75.8	-	-
	Total	100.0			54.8	100.0	54.8

Table A4.55: Reconstituted washability data of the -8+0.5mm size fraction

Size Fraction	RD	Fractional		Cumulative	
		Yield (%)	Ash (%) (DB)	Yield (%)	Ash (%) (DB)
-8+0.5mm	Float 1.30	10.1	5.9	10.1	5.9
	Float 1.35	5.8	11.3	15.9	7.9
	Float 1.40	5.0	15.6	20.8	9.7
	Float 1.45	4.1	19.7	25.0	11.4
	Float 1.50	3.6	24.4	28.5	13.0
	Float 1.55	3.1	29.2	31.6	14.6
	Float 1.60	3.5	33.4	35.1	16.5
	Float 1.65	3.2	37.6	38.3	18.2
	Float 1.70	3.2	41.3	41.5	20.0
	Float 1.75	3.5	45.6	45.0	22.0
	Float 1.80	2.9	48.9	47.9	23.6
	Float 1.85	4.5	53.1	52.4	26.1
	Float 1.90	3.6	57.0	55.9	28.1
	Sink 1.90	44.1	74.7	-	-
	Total	100.0	48.6	100.0	48.6

A4.5.5 Goni Plant Waste Screening and Dense Medium Separation Simulation

Using the reconstituted washability for the $-8+0.5\text{mm}$ size fraction, two simulations were performed, the first to produce a standard Tshikondeni coking coal product of 14% ash, the second was for the production of an alternative product suitable for supply to Eskom at an ash content as close as possible to a typical ash specification level of 30%. The screen was assumed to cut at 8mm with an Ep of 0.250 and the dense medium separation cyclone Ep was set at 0.025, which is within industry norms.

The simulations are illustrated in Figure A4.14 and Figure A4.15.

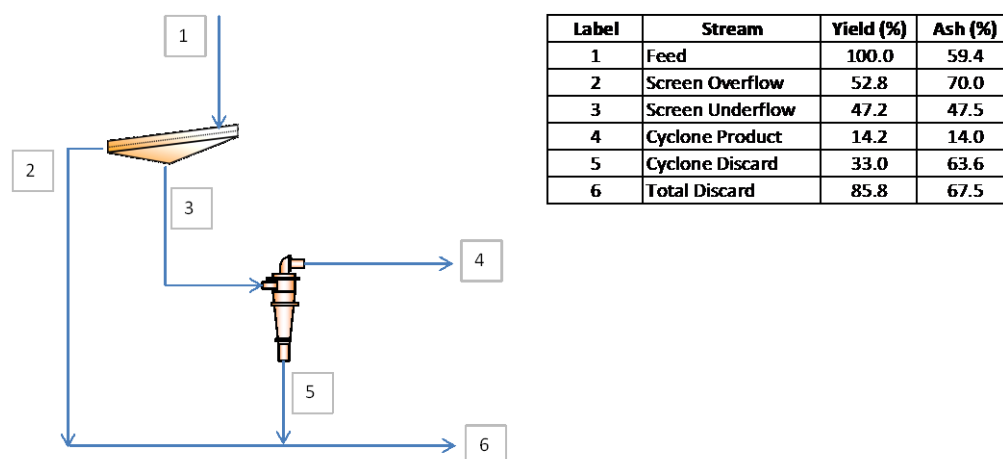


Figure A4.14: Particle size distribution of the sample as produced by the plant

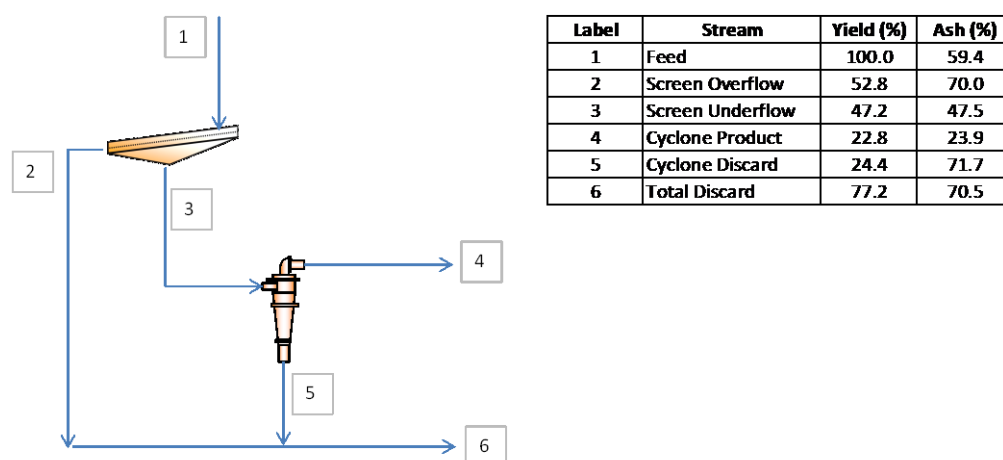


Figure A4.15: Particle size distribution of the sample as produced by the plant

Appendix 5: Laboratory Batch Milling Testwork Results

This Appendix contains the detailed results from the laboratory scale batch milling tests undertaken at Mintek.

A5.1 Laboratory Batch Milling Testwork Results

A5.1.1 Batch Milling Estimated Parameters

The following parameters were estimated by use of the batch ball milling test data for each of the five samples tested, as described in the literature survey section.

Table A5.1: Goni Plant Waste ball mill parameters

Selection function		Breakage function	
kappa	2.0	phi	0.3
alpha	1.5	gamma	0.5
mu	0.8	beta	1.0
lambda	1.5		

Table A5.2: Mutale Plant Waste ball mill parameters

Selection function		Breakage function	
kappa	1.4	phi	0.4
alpha	1.2	gamma	0.5
mu	1.4	beta	1.7
lambda	1.2		

Table A5.3: Mutale Shaft ball mill parameters

Selection function		Breakage function	
kappa	2.3	phi	0.5
alpha	1.4	gamma	1.0
mu	0.8	beta	1.0
lambda	1.5		

Table A5.4: Vhukati Shaft Plant Waste ball mill parameters

Selection function		Breakage function	
kappa	2.4	phi	1.0
alpha	1.5	gamma	0.9
mu	0.8	beta	1.5
lambda	1.5	delta	

Table A5.5: Vhukati Plant Waste ball mill parameters

Selection function		Breakage function	
kappa	2.1	phi	0.7
alpha	1.5	gamma	1.0
mu	0.8	beta	0.9
lambda	1.5	delta	

A5.1.2 Detailed Particle Size Distribution and Material Balance Data

The following tables contain the detailed particle size distribution data and material balance data for the five samples tested. The results are based upon the simulation conducted on a fresh feed tonnage (stream 1) of one tonne per hour entering the circuit, which is illustrated in Circuit flowsheet.

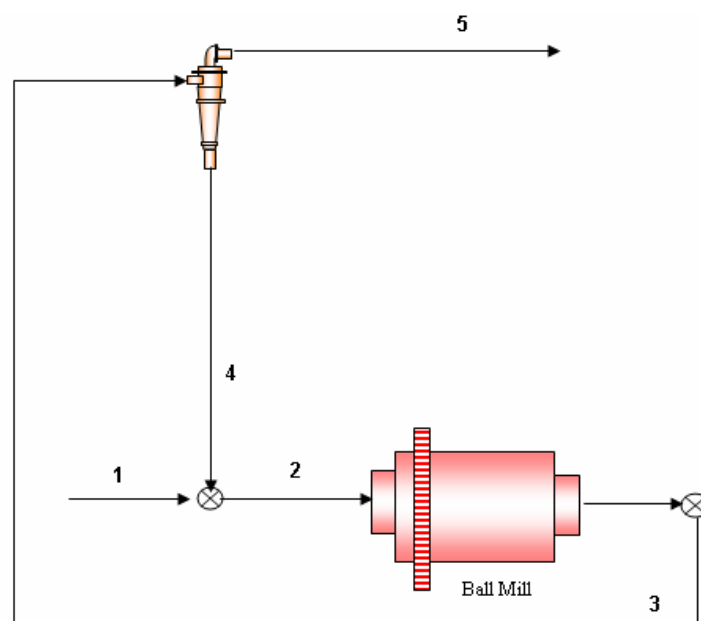


Figure A5.1: Circuit flowsheet

Table A5.6: Goni Plant Waste particle size distributions (percentage passing)

	Stream 1	Stream 2	Stream 3	Stream 4	Stream 5
Size [mm]	New feed	Mill inlet	Mill discharge	Cyc 1. u/f	Cyc 1. o/f
26.500	100.000	100.000	100.00	100.00	100.00
22.400	98.944	99.463	99.94	99.89	100.00
19.000	97.561	98.747	99.84	99.71	100.00
16.000	95.724	97.781	99.70	99.45	100.00
13.200	92.151	95.903	99.42	98.95	100.00
9.500	84.440	91.798	98.78	97.78	100.00
6.700	75.499	86.858	97.84	96.09	100.00
4.750	71.891	84.392	97.00	94.55	100.00
3.350	67.168	81.211	95.93	92.62	100.00
2.360	61.639	77.404	94.60	90.22	100.00
1.700	51.458	70.912	92.68	86.72	100.00
1.180	19.282	51.742	87.91	78.12	99.95
0.85	10.871	43.787	83.60	70.54	99.67
0.6	2.176	34.190	77.35	60.21	98.44
0.425	0.695	27.750	70.22	49.74	95.43
0.3	0.468	21.699	61.57	38.95	89.40
0.212	0.331	16.141	51.86	28.99	80.00
0.15	0.185	11.595	42.14	20.87	68.32
0.106	0.106	8.240	33.39	14.85	56.20
0.075	0.053	5.903	26.12	10.66	45.16
0.053	0.026	4.295	20.33	7.76	35.79
0.038	0.014	3.256	16.03	5.89	28.51
Solids (tph)	1.000	2.231	2.231	1.231	1.000

Table A5.7: Goni Plant Waste material balance (flows in t/h)

	Stream 1	Stream 2	Stream 3	Stream 4	Stream 5
Size [mm]	New feed	Mill inlet	Mill discharge	Cyc 1. u/f	Cyc 1. o/f
26.500	0.011	0.012	0.001	0.001	0.000
22.400	0.014	0.016	0.002	0.002	0.000
19.000	0.018	0.022	0.003	0.003	0.000
16.000	0.036	0.042	0.006	0.006	0.000
13.200	0.077	0.092	0.014	0.014	0.000
9.500	0.089	0.110	0.021	0.021	0.000
6.700	0.036	0.055	0.019	0.019	0.000
4.750	0.047	0.071	0.024	0.024	0.000
3.350	0.055	0.085	0.030	0.030	0.000
2.360	0.102	0.145	0.043	0.043	0.000
1.700	0.322	0.428	0.106	0.106	0.001
1.180	0.084	0.177	0.096	0.093	0.003
0.85	0.087	0.214	0.140	0.127	0.012
0.6	0.015	0.144	0.159	0.129	0.030
0.425	0.002	0.135	0.193	0.133	0.060
0.3	0.001	0.124	0.217	0.123	0.094
0.212	0.001	0.101	0.217	0.100	0.117
0.15	0.001	0.075	0.195	0.074	0.121
0.106	0.001	0.052	0.162	0.052	0.110
0.075	0.000	0.036	0.129	0.036	0.094
0.053	0.000	0.023	0.096	0.023	0.073
0.038	0.000	0.072	0.357	0.072	0.285
Solids (tph)	1.000	2.231	2.231	1.231	1.000

Table A5.8: Mutale Plant Waste particle size distributions (percentage passing)

	Stream 1	Stream 2	Stream 3	Stream 4	Stream 5
Size [mm]	New feed	Mill inlet	Mill discharge	Cyc 1. u/f	Cyc 1. o/f
26.500	100.000	100.000	100.00	100.00	100.00
22.400	99.413	99.712	99.98	99.97	100.00
19.000	97.241	98.642	99.91	99.83	100.00
16.000	93.178	96.621	99.75	99.54	100.00
13.200	87.127	93.572	99.48	99.04	100.00
9.500	75.661	87.702	98.87	97.91	100.00
6.700	60.312	79.651	97.86	96.05	100.00
4.750	54.190	75.774	96.79	94.08	100.00
3.350	47.102	71.157	95.43	91.55	100.00
2.360	40.378	66.352	93.71	88.38	100.00
1.700	32.502	60.548	91.52	84.33	99.99
1.180	13.302	47.764	87.51	76.99	99.92
0.85	7.348	40.708	83.03	69.00	99.58
0.6	1.286	32.182	76.67	58.38	98.25
0.425	0.288	25.648	69.15	47.15	95.09
0.3	0.172	19.470	60.26	35.83	89.07
0.212	0.114	13.969	50.63	25.72	80.00
0.15	0.064	9.624	41.20	17.73	68.88
0.106	0.040	6.523	32.79	12.02	57.30
0.075	0.024	4.467	25.89	8.23	46.70
0.053	0.013	3.135	20.43	5.78	37.71
0.038	0.007	2.343	16.42	4.32	30.69
Solids (tph)	1.000	2.179	2.179	1.179	1.000

Table A5.9: Mutale Plant Waste material balance (flows in t/h)

	Stream 1	Stream 2	Stream 3	Stream 4	Stream 5
Size [mm]	New feed	Mill inlet	Mill discharge	Cyc 1. u/f	Cyc 1. o/f
26.500	0.006	0.006	0.000	0.000	0.000
22.400	0.022	0.023	0.002	0.002	0.000
19.000	0.041	0.044	0.003	0.003	0.000
16.000	0.061	0.066	0.006	0.006	0.000
13.200	0.115	0.128	0.013	0.013	0.000
9.500	0.153	0.175	0.022	0.022	0.000
6.700	0.061	0.084	0.023	0.023	0.000
4.750	0.071	0.101	0.030	0.030	0.000
3.350	0.067	0.105	0.037	0.037	0.000
2.360	0.079	0.126	0.048	0.048	0.000
1.700	0.192	0.279	0.087	0.087	0.001
1.180	0.060	0.154	0.098	0.094	0.003
0.85	0.061	0.186	0.138	0.125	0.013
0.6	0.010	0.142	0.164	0.132	0.032
0.425	0.001	0.135	0.194	0.133	0.060
0.3	0.001	0.120	0.210	0.119	0.091
0.212	0.001	0.095	0.205	0.094	0.111
0.15	0.000	0.068	0.183	0.067	0.116
0.106	0.000	0.045	0.151	0.045	0.106
0.075	0.000	0.029	0.119	0.029	0.090
0.053	0.000	0.017	0.087	0.017	0.070
0.038	0.000	0.051	0.358	0.051	0.307
Solids (tph)	1.000	2.179	2.179	1.179	1.000

Table A5.10: Mutale Shaft particle size distributions (percentage passing)

	Stream 1	Stream 2	Stream 3	Stream 4	Stream 5
Size [mm]	New feed	Mill inlet	Mill discharge	Cyc 1. u/f	Cyc 1. o/f
26.500	100.000	100.000	100.00	100.00	100.00
22.400	97.993	99.044	99.85	99.76	100.00
19.000	94.276	97.247	99.56	99.27	100.00
16.000	87.581	93.986	99.01	98.34	100.00
13.200	80.433	90.394	98.32	97.17	100.00
9.500	64.645	82.427	96.74	94.53	100.00
6.700	50.978	75.006	94.86	91.36	100.00
4.750	44.777	70.706	93.07	88.35	100.00
3.350	39.086	66.412	91.08	85.01	100.00
2.360	34.200	62.237	88.88	81.32	100.00
1.700	27.534	57.020	86.36	77.09	99.99
1.180	11.307	46.430	82.32	70.33	99.93
0.85	7.033	40.796	78.29	63.77	99.63
0.6	1.533	33.433	72.68	55.14	98.45
0.425	0.604	27.542	65.98	45.88	95.53
0.3	0.424	21.566	57.68	35.95	89.59
0.212	0.308	15.842	48.11	26.41	80.00
0.15	0.183	10.992	38.28	18.35	67.57
0.106	0.107	7.346	29.25	12.27	54.19
0.075	0.053	4.835	21.75	8.09	41.82
0.053	0.024	3.172	15.87	5.32	31.39
0.038	0.010	2.168	11.66	3.64	23.45
Solids (tph)	1.000	2.469	2.469	1.469	1.000

Table A5.11: Mutale Shaft material balance (flows in t/h)

Size [mm]	Stream 1 New feed	Stream 2 Mill inlet	Stream 3 Mill discharge	Stream 4 Cyc 1. u/f	Stream 5 Cyc 1. o/f
26.500	0.020	0.024	0.004	0.004	0.000
22.400	0.037	0.044	0.007	0.007	0.000
19.000	0.067	0.081	0.014	0.014	0.000
16.000	0.071	0.089	0.017	0.017	0.000
13.200	0.158	0.197	0.039	0.039	0.000
9.500	0.137	0.183	0.047	0.047	0.000
6.700	0.062	0.106	0.044	0.044	0.000
4.750	0.057	0.106	0.049	0.049	0.000
3.350	0.049	0.103	0.054	0.054	0.000
2.360	0.067	0.129	0.062	0.062	0.000
1.700	0.162	0.262	0.100	0.099	0.001
1.180	0.043	0.139	0.099	0.096	0.003
0.85	0.055	0.182	0.139	0.127	0.012
0.6	0.009	0.145	0.165	0.136	0.029
0.425	0.002	0.148	0.205	0.146	0.059
0.3	0.001	0.141	0.236	0.140	0.096
0.212	0.001	0.120	0.243	0.118	0.124
0.15	0.001	0.090	0.223	0.089	0.134
0.106	0.001	0.062	0.185	0.061	0.124
0.075	0.000	0.041	0.145	0.041	0.104
0.053	0.000	0.025	0.104	0.025	0.079
0.038	0.000	0.054	0.288	0.053	0.234
Solids (tph)	1.000	2.469	2.469	1.469	1.000

Table A5.12: Vhukati Shaft particle size distributions (percentage passing)

	Stream 1	Stream 2	Stream 3	Stream 4	Stream 5
Size [mm]	New feed	Mill inlet	Mill discharge	Cyc 1. u/f	Cyc 1. o/f
26.500	100.000	100.000	100.00	100.00	100.00
22.400	98.141	99.094	99.89	99.81	100.00
19.000	94.018	97.061	99.62	99.34	100.00
16.000	86.403	93.277	99.10	98.43	100.00
13.200	77.271	88.635	98.37	97.15	100.00
9.500	59.928	79.673	96.84	94.47	100.00
6.700	44.658	71.247	94.95	91.17	100.00
4.750	39.811	67.459	93.24	88.18	100.00
3.350	33.268	62.631	91.22	84.64	100.00
2.360	28.265	58.248	88.98	80.72	100.00
1.700	22.729	53.370	86.47	76.33	100.00
1.180	10.840	44.528	82.70	69.77	99.94
0.85	5.996	38.516	78.64	62.89	99.66
0.6	1.314	31.406	73.06	53.96	98.54
0.425	0.443	25.441	66.24	44.17	95.69
0.3	0.281	19.380	57.70	33.69	89.73
0.212	0.187	13.667	47.86	23.77	80.00
0.15	0.097	8.993	37.82	15.66	67.38
0.106	0.056	5.655	28.72	9.85	53.90
0.075	0.028	3.491	21.27	6.09	41.53
0.053	0.013	2.146	15.50	3.75	31.18
0.038	0.007	1.398	11.39	2.44	23.33
Solids (tph)	1.000	2.334	2.334	1.334	1.000

Table A5.13: Vhukati Shaft material balance (flows in t/h)

	Stream 1	Stream 2	Stream 3	Stream 4	Stream 5
Size [mm]	New feed	Mill inlet	Mill discharge	Cyc 1. u/f	Cyc 1. o/f
26.500	0.019	0.021	0.003	0.003	0.000
22.400	0.041	0.047	0.006	0.006	0.000
19.000	0.076	0.088	0.012	0.012	0.000
16.000	0.091	0.108	0.017	0.017	0.000
13.200	0.173	0.209	0.036	0.036	0.000
9.500	0.153	0.197	0.044	0.044	0.000
6.700	0.048	0.088	0.040	0.040	0.000
4.750	0.065	0.113	0.047	0.047	0.000
3.350	0.050	0.102	0.052	0.052	0.000
2.360	0.055	0.114	0.059	0.059	0.000
1.700	0.119	0.206	0.088	0.088	0.001
1.180	0.048	0.140	0.095	0.092	0.003
0.85	0.047	0.166	0.130	0.119	0.011
0.6	0.009	0.139	0.159	0.130	0.029
0.425	0.002	0.141	0.199	0.140	0.060
0.3	0.001	0.133	0.230	0.132	0.097
0.212	0.001	0.109	0.234	0.108	0.126
0.15	0.000	0.078	0.212	0.077	0.135
0.106	0.000	0.050	0.174	0.050	0.124
0.075	0.000	0.031	0.135	0.031	0.104
0.053	0.000	0.017	0.096	0.017	0.078
0.038	0.000	0.033	0.266	0.032	0.233
Solids (tph)	1.000	2.334	2.334	1.334	1.000

Table A5.14: Vhukati Plant Waste Shaft particle size distributions (percentage passing)

	Stream 1	Stream 2	Stream 3	Stream 4	Stream 5
Size [mm]	New feed	Mill inlet	Mill discharge	Cyc 1. u/f	Cyc 1. o/f
26.500	100.000	100.000	100.00	100.00	100.00
22.400	99.896	99.947	99.99	99.99	100.00
19.000	99.712	99.853	99.98	99.97	100.00
16.000	99.478	99.732	99.97	99.94	100.00
13.200	99.070	99.520	99.94	99.89	100.00
9.500	91.167	95.528	99.52	99.12	100.00
6.700	80.181	89.799	98.75	97.72	100.00
4.750	76.005	87.220	98.06	96.46	100.00
3.350	69.344	83.237	97.08	94.68	100.00
2.360	61.987	78.646	95.82	92.37	100.00
1.700	47.870	70.279	93.82	88.74	100.00
1.180	18.641	53.016	89.73	81.33	99.93
0.85	8.788	44.525	85.55	73.97	99.62
0.6	2.205	36.349	79.83	64.48	98.46
0.425	0.473	29.782	72.73	53.93	95.55
0.3	0.264	23.431	63.80	42.52	89.63
0.212	0.164	17.273	53.34	31.37	80.00
0.15	0.101	12.039	42.49	21.87	67.53
0.106	0.067	8.086	32.51	14.69	54.13
0.075	0.048	5.363	24.22	9.74	41.80
0.053	0.030	3.547	17.73	6.45	31.43
0.038	0.019	2.444	13.07	4.44	23.54
Solids (tph)	1.000	2.214	2.214	1.214	1.000

Table A5.15: Vhukati Plant Waste material balance (flows in t/h)

Size [mm]	Stream 1 New feed	Stream 2 Mill inlet	Stream 3 Mill discharge	Stream 4 Cyc 1. u/f	Stream 5 Cyc 1. o/f
26.500	0.001	0.001	0.000	0.000	0.000
22.400	0.002	0.002	0.000	0.000	0.000
19.000	0.002	0.003	0.000	0.000	0.000
16.000	0.004	0.005	0.001	0.001	0.000
13.200	0.079	0.088	0.009	0.009	0.000
9.500	0.110	0.127	0.017	0.017	0.000
6.700	0.042	0.057	0.015	0.015	0.000
4.750	0.067	0.088	0.022	0.022	0.000
3.350	0.074	0.102	0.028	0.028	0.000
2.360	0.141	0.185	0.044	0.044	0.000
1.700	0.292	0.382	0.091	0.090	0.001
1.180	0.099	0.188	0.092	0.089	0.003
0.85	0.066	0.181	0.127	0.115	0.012
0.6	0.017	0.145	0.157	0.128	0.029
0.425	0.002	0.141	0.198	0.138	0.059
0.3	0.001	0.136	0.232	0.135	0.096
0.212	0.001	0.116	0.240	0.115	0.125
0.15	0.000	0.087	0.221	0.087	0.134
0.106	0.000	0.060	0.183	0.060	0.123
0.075	0.000	0.040	0.144	0.040	0.104
0.053	0.000	0.024	0.103	0.024	0.079
0.038	0.000	0.054	0.290	0.054	0.235
Solids (tph)	1.000	2.214	2.214	1.214	1.000

Appendix 6: 20 Tonne Bulk Sample Flotation Test Results

This appendix contains the data for Chapter 6 of the thesis from the various flotation tests performed on the 20 tonne bulk sample extracted from an readily accessible section of the old dump.

From the screening section of the work it was evident that little value was contained in the coarser size fractions, therefore work on this sample was conducted on the material in the dump at 6mm, 3mm and 0.5mm.

Samples were also subjected to a destoning using single density float and sinks tests. The effect of using organic liquids compared to a magnetite-water suspension was also examined.

A6.1 Separation Using a Magnetite Suspension

This section examines the effect of destoning by size at 3mm and 6mm respectively as well as by a single density float and sinks test. The separation density varied due to the difficulty of replicating the tests using a water and magnetite suspension as detailed in the relevant section.

A6.1.1 Tests Performed on -3mm Material

The samples were screened at 3mm and 0.5mm. The -3+0.5mm fraction as upgraded by single density float and sinks, the floats fraction added back to the -0.5mm fraction prior to milling to -212µm. The three rougher rate test results are presented in Table A6.1:

Table A6.1: Rougher rate flotation tests

		150 g/t FTN4				
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
-3mm Discard F1.75	Conc 1	0.50	23.5	23.5	16.5	16.5
	Conc 2	1.00	8.5	32.0	15.8	16.4
	Conc 3	2.00	11.4	43.4	17.6	16.7
	Tails	2.00	56.6	-	30.0	-
	Feed		100.0	100.0	24.2	24.2
		150 g/t Par , 150 g/t FTN4				
Sample	Stream	Time (min)	%Mass Yield	Cum. Yield	%Ash	Cum. % Ash
-3mm Discard F1.75	Conc 1	0.50	24.8	24.8	17.1	17.1
	Conc 2	1.00	13.1	37.9	17.6	17.3
	Conc 3	2.00	12.4	50.3	22.7	18.6
	Tails	2.00	49.7	-	29.9	-
	Feed		100.0	100.0	24.2	24.2
		150 g/t Par , 150 g/t FTN4 , 300g/t Cytec S-8860 GL				
Sample	Stream	Time (min)	%Mass Yield	Cum. Yield	%Ash	Cum. % Ash
-3mm Discard F1.75	Conc 1	0.50	5.2	5.2	16.1	16.1
	Conc 2	1.00	3.1	8.4	14.4	15.5
	Conc 3	2.00	5.7	14.1	15.4	15.4
	Tails	2.00	85.9	100.0	25.6	-
	Feed		100.0	100.0	24.2	24.2

A6.1.2 Tests Performed on -6mm Material

The material was screened at 6mm and 0.5mm, the -6+0.5mm floats fraction added back to the -0.5mm fraction prior to milling to -212µm. Three rougher rate tests were performed; thereafter a cleaner rate test was performed upon the combined rougher concentrates from these tests. The rougher concentrates were also milled for one minute without water to give a basic regrind and with the view to improve liberation of values. These results are presented in Table A6.2 and Table A6.3.

Table A6.2: Rougher rate flotation tests

100 g/t Par , 50 g/t FTN4						
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
-6mm Discard F1.68	Conc 1	0.50	6.2	6.2	20.5	20.5
	Conc 2	1.00	9.3	15.5	22.3	21.6
	Conc 3	2.00	12.6	28.1	22.8	22.1
	Conc 4	4.00	13.4	41.4	27.5	23.9
	Tails	4.00	58.6	-	33.4	-
	Feed		100.0	100.0	29.4	29.5
100 g/t Par , 50 g/t FTN4 , 100g/t Cytec S-8860 GL						
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
-6mm Discard F1.68	Conc 1	0.50	5.0	5.0	19.4	19.4
	Conc 2	1.00	3.7	8.7	19.1	19.3
	Conc 3	2.00	8.1	16.8	21.7	20.5
	Conc 4	4.00	8.9	25.7	22.6	21.2
	Tails	4.00	74.3	-	32.7	-
	Feed		100.0	100.0	29.7	29.7
150 g/t Par , 50 g/t FTN4 , 100g/t Cytec S-8860 GL						
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
-6mm Discard F1.68	Conc 1	0.50	4.7	4.7	18.3	18.3
	Conc 2	1.00	4.6	9.3	19.1	18.7
	Conc 3	2.00	11.3	20.6	23.7	21.4
	Conc 4	4.00	9.7	30.3	23.6	22.1
	Tails	4.00	69.7	-	32.6	-
	Feed		100.0	100.0	29.4	29.4

Table A6.3: Cleaner rate flotation test

		200g/t CRX11 and 100g/t FTN4				
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
-6mm Discard F1.68	Conc 1	0.50	10.7 (3.5)	10.7 (3.5)	18.0	18.0
	Conc 2	1.00	8.4 (2.7)	19.1 (6.2)	18.4	18.1
	Conc 3	2.00	23.6 (7.7)	42.7 (13.9)	20.3	19.3
	Tails	4.00.	57.3	-	37.0	-
	Feed		100.0	100.0	23.3	23.3

The figures in brackets show the concentrate yield relative to the rougher feed.

A6.2 Discard As Received

The whole discard was screened at 0.5mm material was subjected to the float and sinks separation followed by the floats F1.72 fraction added back to the -0.5mm material and milled to -212µm.

A6.2.1 Rougher Rate Tests on Floats F1.72 Material

The results are presented in Table A6.4

Table A6.4: Rougher rate flotation tests

100 g/t Par , 100 g/t FTN4 (OLD CELL)						
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard As Received F1.72	Conc 1	0.50	14.9	14.9	23.6	23.6
	Conc 2	1.00	11.5	26.3	23.4	23.5
	Conc 3	2.00	9.9	36.2	23.9	23.6
	Conc 4	4.00	12.0	48.2	28.3	24.8
	Tails	4.00	51.8	-	33.4	-
	Feed		100.0	100.0	29.3	29.3
200 g/t CRX 11						
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard As Received F1.72	Conc 1	0.50	13.1	13.1	24.0	24.0
	Conc 2	1.00	8.4	21.5	24.0	24.0
	Conc 3	2.00	8.5	29.9	24.9	24.2
	Conc 4	4.00	10.5	40.4	26.1	24.7
	Tails	4.00	59.6	-	32.2	-
	Feed		100.0	100.0	29.2	29.2
400 g/t CRX 11 and 100g/t FTN4						
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard As Received F1.72	Conc 1	0.50	25.6	25.6	24.8	24.8
	Conc 2	1.00	11.6	37.2	25.3	25.0
	Conc 3	2.00	8.4	45.6	26.1	25.2
	Conc 4	4.00.	3.5	49.1	27.1	25.3
	Tails	4.00.	50.9	-	33.4	-
	Feed		100.0	100.0	29.4	29.4
200 g/t CRX 11 and 200g/t FTN4						
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard As Received F1.72	Conc 1	0.50	35.3	35.3	25.6	25.6
	Conc 2	1.00	11.9	47.2	25.6	25.6
	Conc 4	4.00.	6.8	54.0	26.5	25.7
	Tails	4.00.	46.0	-	33.3	-
	Feed		100.0	100.0	29.2	29.2

A6.2.2 Bulk Rougher Flotation Test of Floats 1.65

The whole discard was screened at 0.5mm material was subjected to the float and sinks separation followed by the floats F1.65 fraction added back to the -0.5mm material and milled to -212µm.

Table A6.5: Bulk rougher flotation test

		200 g/t CRX 11+400g/t FTN4				
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard As Rec'd F1.65	Conc 1	4.00.	38.84	38.84	19.47	19.47
	Tails	4.00.	61.16	-	27.52	-
	Feed		100.00	100.00	24.39	24.39

A6.3 Discard -0.5mm Size Fraction

This work was performed on the -0.5mm material to replicate what could be achieved by flotation from this size fraction

A6.3.1 Bulk Rougher Tests Examining the Effect of Milling to 212µm

These two tests show the flotation performance of the -0.5mm fraction “as is: and after milling to -212µm

Table A6.6: Bulk rougher flotation tests

		200 g/t CRX 11+400g/t FTN4 and 200g/t PAR				
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard -0.5mm	Conc 1	4.00	44.1	44.1	20.1	20.1
	Tails	4.00	55.9	-	59.7	42.3
	Feed		100.0	100.0	42.3	42.3
		200 g/t CRX 11+400g/t FTN4 and 200g/t PAR				
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard Milled to -212µm	Conc 1	4.00	69.2	69.2	27.8	27.8
	Tails	4.00	30.8	-	66.7	-
	Feed		100.0	100.0	39.7	39.7

A6.3.2 Rougher Tests on -0.5mm Material

This set of tests examined the effect of different reagent dosages upon the -.0.5mm material

Table A6.7: Rougher rate test results

100 g/t CRX 11+100g/t FTN4						
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard -0.5mm	Conc 1	0.50	18.2	18.2	13.9	13.9
	Conc 2	1.00	8.5	26.7	15.4	14.3
	Conc 3	2.00	6.6	33.2	19.3	15.3
	Tails	2.00.	66.8	-	52.7	-
	Feed		100.0	100.0	40.2	40.2
200 g/t CRX 11+100g/t FTN4						
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard -0.5mm	Conc 1	0.50	22.9	22.9	15.2	15.2
	Conc 2	1.00	10.3	33.1	15.3	15.2
	Conc 4	2.00	6.8	39.9	21.8	16.3
	Tails	2.00.	60.1	-	57.4	-
	Feed		100.0	100.0	41.0	41.0

A6.3.3 Bulk Rougher Comparative Tests Using 100g/t CRX11 and 100g/t FTN4

These seven tests were undertaken to examine repeatability of the tests under identical conditions.

Table A6.8: Bulk rougher flotation test results

100 g/t CRX 11+100g/t FTN4						
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard -0.5mm Test 1	Conc 1	4.00	40.8	40.8	18.0	18.0
	Tails	4.00	59.2	-	56.3	-
	Feed		100.0	100.0	40.7	40.7
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard -0.5mm Test 2	Conc 1	4.00	36.1	36.1	18.3	18.3
	Tails	4.00	63.9	-	52.6	-
	Feed		100.0	100.0	40.2	40.2
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard -0.5mm Test 3	Conc 1	4.00	38.7	38.7	16.5	16.5
	Tails	4.00	61.3	-	56.4	-
	Feed		100.0	100.0	41.0	41.0
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard -0.5mm Test 4	Conc 1	4.00	37.9	37.9	16.4	16.4
	Tails	4.00	62.1	-	55.8	-
	Feed		100.0	100.0	40.8	40.8
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard -0.5mm Test 5	Conc 1	4.00	41.2	41.2	17.3	17.3
	Tails	4.00	58.8	-	59.5	-
	Feed		100.0	100.0	42.1	42.1
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard -0.5mm Test 6	Conc 1	4.00	40.8	40.8	17.0	17.0
	Tails	4.00	59.2	-	56.1	-
	Feed		100.0	100.0	40.2	40.2
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard -0.5mm Test 7	Conc 1	4.00	39.3	39.3	16.9	16.9
	Tails	4.00	60.7	-	56.1	-
	Feed		100.0	100.0	40.7	40.7

A6.3.4 Bulk Rougher Comparative Tests Using 100g/t Paraffin and 100g/t FTN4

These samples were performed using paraffin not CRX11 as the collector for repeatability testing. Thereafter, the concentrates were used for the next section to examine the effect of a cleaner stage of flotation.

Table A6.9: Bulk rougher flotation test results

		100 g/t PAR 11+100g/t FTN4				
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard -0.5mm Test 1	Conc 1	4.00	34.4	34.4	17.1	17.1
	Tails	4.00	65.6	-	54.5	-
	Feed		100.0	100.0	41.6	41.7
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard -0.5mm Test 2	Conc 1	4.00	25.5	25.5	17.8	17.8
	Tails	4.00	74.5	-	50.1	-
	Feed		100.0	100.0	41.9	41.9
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard -0.5mm Test 3	Conc 1	4.00	26.2	26.2	15.0	15.0
	Tails	4.00	73.8	-	50.0	-
	Feed		100.0	100.0	40.8	40.8
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Discard -0.5mm Test 4	Conc 1	4.00	27.0	27.0	15.5	15.5
	Tails	4.00	73.0	-	49.2	-
	Feed		100.0	100.0	40.1	40.1

A6.3.5 Bulk Cleaner Tests Using 100g/t Paraffin and 100g/t FTN4 in the Rougher Stage

The concentrates from the previous section were used to perform two pairs of tests. The first pair using different milling times of 1 minute and 2 minutes respectively to examine the effect of a regrind milling stage ahead of cleaner flotation. The second pair did not consider milling but looked at using 50% less collector and frother in the second test relative to the first.

The results are presented in Table A6.10 to Table A6.13.

Table A6.10: Bulk cleaner test - pair 1, test 1

		100 g/t PAR 11+100g/t FTN4				
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Milled 1 Minute	Conc 1	4.00	54.1	54.1	13.2	13.2
	Tails	4.00	45.9	-	21.0	-
	Feed		100.0	100.0	16.8	16.8

For this above test, the overall yield relative to the rougher stage was determined to be 18.6%.

Table A6.11: Bulk cleaner test – pair 1, test 2

		100 g/t PAR+100g/t FTN4				
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Milled 2 Minutes	Conc 1	4.00	47.7	47.7	13.6	13.6
	Tails	4.00	52.3	-	19.6	-
	Feed		100.0	100.0	16.7	16.7

For this above test, the overall yield relative to the rougher stage was determined to be 12.2%.

Table A6.12: Bulk cleaner test – pair 2, test 1

		50 g/t PAR 11+50g/t FTN4				
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Not Milled	Conc 1	4.00	80.5	80.5	13.9	13.9
	Tails	4.00	19.5	-	26.8	-
	Feed		100.0	100.0	16.4	16.4

Appendix 6: 20 Tonne Bulk Sample Flotation Test Results

For this above test, the overall yield relative to the rougher stage was determined to be 21.1%.

Table A6.13: Bulk cleaner test – pair 2, test 2

		100 g/t PAR+100g/t FTN4				
Sample	Stream	Time (min)	Yield (%)	Cum Yield (%)	Ash (%)	Cum Ash (%)
Not Milled	Conc 1	4.00	78.4	78.4	14.0	14.0
	Tails	4.00	21.6	-	26.8	-
	Feed		100.0	100.0	16.7	16.7

For this above test, the overall yield relative to the rougher stage was determined to be 22.2%.

Appendix 7: Financial Evaluation

This appendix contains the data related to the detailed working costs used in Chapter 8 for the financial evaluation of the plant.

12.2 Detailed Working Costs Estimates Used in the Financial Evaluation

The detailed working cost estimates used in Chapter 8 are summarized in Table A7.1.

Table A7.1: Working costs estimates

Apect of Project	Unit	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Opex variable (FOR)	R/t																	
Utilization	%	80	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00
Day	hrs/day	24	24	24	24	24	24	24	24	24	24	24	24	24	24	24	24	24
Year	days/yr	365	365	365	365	365	365	365	365	365	365	365	365	365	365	365	365	365
Hours	hrs/yr	7008	7008	7008	7008	7008	7008	7008	7008	7008	7008	7008	7008	7008	7008	7008	7008	7008
Feedstock	t/hr	80	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00	80.00
Feedstock / year	tpa	560640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640
Cost of Feedstock	R/ton	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Feedstock Cost	R/yr	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0	R 0
Electricity	kWh	150	150	150	150	150	150	150	150	150	150	150	150	150	150	150	150	150
Electricity/yr	kW/yr	1 051 200	1 051 200	1 051 200	1 051 200	1 051 200	1 051 200	1 051 200	1 051 200	1 051 200	1 051 200	1 051 200	1 051 200	1 051 200	1 051 200	1 051 200	1 051 200	1 051 200
Cost	R/kW	0.55	0.69	0.87	0.99	1.05	1.10	1.16	1.21	1.28	1.34	1.41	1.48	1.55	1.63	1.71	1.79	1.88
Total Cost	R/yr	R 578 160	R 729 927	R 912 409	R 1 037 865	R 1 102 732	R 1 157 868	R 1 215 761	R 1 276 550	R 1 340 377	R 1 407 396	R 1 477 766	R 1 551 654	R 1 629 237	R 1 710 699	R 1 796 233	R 1 886 045	R 1 980 347
Water	m3/hr	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80
Water/yr	m3/yr	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640	560 640
Cost	R/m3	7.6	8.25	8.66	9.09	9.55	10.02	10.52	11.05	11.60	12.18	12.79	13.43	14.10	14.81	15.55	16.33	17.14
Total Cost	R/yr	R 4 260 864	R 4 623 037	R 4 854 189	R 5 096 899	R 5 351 744	R 5 619 331	R 5 900 297	R 6 195 312	R 6 505 078	R 6 830 332	R 7 171 848	R 7 530 441	R 7 906 963	R 8 302 311	R 8 717 427	R 9 153 298	R 9 610 963
Maintenance	R/yr	R 655 401	R 701 279	R 739 850	R 776 842	R 815 684	R 856 468	R 899 292	R 944 256	R 991 469	R 1 041 043	R 1 093 095	R 1 147 750	R 1 205 137	R 1 265 394	R 1 328 664	R 1 395 097	R 1 464 852
Transport and Harbour cost	R/t	R 90	R 97	R 103	R 108	R 114	R 119	R 125	R 131	R 138	R 145	R 152	R 160	R 168	R 176	R 185	R 194	R 204
Total Cost	R/yr	R 5 045 760	R 5 449 421	R 5 776 386	R 6 065 205	R 6 368 466	R 6 686 889	R 7 021 233	R 7 372 295	R 7 740 910	R 8 127 955	R 8 534 353	R 8 961 071	R 9 409 124	R 9 879 580	R 10 373 559	R 10 892 237	R 11 436 849
Product																		
Yield	%	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10
Product	tpa	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064	56 064
Price	\$/t	\$120	\$137	\$154	\$180	\$149	\$137	\$129	\$128	\$128	\$128	\$128	\$128	\$128	\$128	\$128	\$128	\$128
Price	R/t	R 1 080	R 1 249	R 1 413	R 1 671	R 1 420	R 1 343	R 1 311	R 1 335	R 1 374	R 1 414	R 1 456	R 1 498	R 1 543	R 1 588	R 1 631	R 1 683	R 1 733
Total Price	R/yr	R 60 549 120	R 70 025 035	R 79 235 396	R 93 707 979	R 79 613 178	R 75 317 045	R 73 482 545	R 74 821 408	R 77 045 246	R 79 269 085	R 81 636 397	R 84 003 709	R 86 514 495	R 89 025 280	R 91 464 329	R 94 333 798	R 97 131 530
Assumptions, 2009 terms, ZAR																		
Capital																		
Project capital (including 20% contingency)	R	R 21 846 704	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sustaining capital	%	0%	0%	0%	0%	0%	0%	0%	2%	2%	2%	2%	2%	2%	2%	2%	2%	2%
Opex																		
Fixed	R	-	-	-	-	-	-	-	944 256	991 469	1 041 043	1 093 095	1 147 750	1 205 137	1 265 394	1 328 664	1 395 097	1 464 852
Variable	R/t product	-	-	-	-	-	-	-	110.50	116.03	121.83	127.92	134.32	141.03	148.09	155.49	163.27	171.43
Electricity	R/t product	-	-	-	-	-	-	-	23	24	25	26	28	29	31	32	34	35
Transport and Harbour cost	R/t product	-	-	-	-	-	-	-	131	138	145	152	160	168	176	185	194	204
Production																		
Hard Coking Coal	tpa	-	-	-	-	-	-	-	56 064.00	56 064.00	56 064.00	56 064.00	56 064.00	56 064.00	56 064.00	56 064.00	56 064.00	56 064.00

Appendix 8: Fines DMS Plant

This appendix contains the data related to the installation of a fines DMS circuit into what was the decommissioned spirals section of the plant which was discussed in Chapter 9 of the main body of the thesis.

A8.1 Screening and Washability Data

This section details the screening and washability data from samples of plant feed taken in March 2007.

A8.1.1 Screening Data

The screening results for Mutale Shaft and Vhukati Shaft are shown in presented in Table A8.1 and Table A8.2 respectively

Table A8.1: Mutale Shaft particle size distribution

Top size (mm)	Bottom size (mm)	Mass (g)	Mass Retained (%)	Cumulative Retained (%)	Cumulative Passing (%)
	13.2	3027	14.1	14.1	85.9
13.2	9.5	2440	11.4	25.5	74.5
9.5	6.7	2980	13.9	39.4	60.6
6.7	4.75	2159	10.1	49.5	50.5
4.75	3.35	1334	6.2	55.7	44.3
3.35	2.36	1589	7.4	63.1	36.9
2.36	1.7	1232	5.7	68.9	31.1
1.7	1.4	597	2.8	71.7	28.3
1.4	0.85	1620	7.6	79.2	20.8
0.85	0.6	786	3.7	82.9	17.1
0.6	0.425	814	3.8	86.7	13.3
0.425	0.3	690	3.2	89.9	10.1
0.3	0.212	906	4.2	94.1	5.9
0.212	0.15	884	4.1	98.3	1.7
0.15	0.106	254	1.2	99.4	0.6
0.106	0.075	109	0.5	99.9	0.0
0.075	0	11	0.1	-	-
Total		21431	100.0	100.0	0.0

Table A8.2: Vhukati Shaft particle size distribution

Top size (mm)	Bottom size (mm)	Mass (g)	Mass Retained (%)	Cumulative Retained (%)	Cumulative Passing (%)
	13.2	2041	11.4	11.4	88.6
13.2	9.5	1713	9.6	20.9	79.1
9.5	6.7	2224	12.4	33.3	66.7
6.7	4.75	1557	8.7	42.0	58.0
4.75	3.35	1130	6.3	48.3	51.7
3.35	2.36	1365	7.6	55.9	44.1
2.36	1.7	1143	6.4	62.3	37.7
1.7	1.4	602	3.4	65.7	34.3
1.4	0.85	1734	9.7	75.3	24.7
0.85	0.6	716	4.0	79.3	20.7
0.6	0.425	706	3.9	83.3	16.7
0.425	0.3	616	3.4	86.7	13.3
0.3	0.212	682	3.8	90.5	9.5
0.212	0.15	1012	5.6	96.2	3.8
0.15	0.106	389	2.2	98.3	1.7
0.106	0.075	246	1.4	99.7	0.3
0.075	0	53	0.3	-	-
Total		17929	100.0	100.0	0.0

A8.1.2 Washability Data

The results are presented in Table A8.3 and Table A8.4 for Mutale Shaft and Vhukati Shaft respectively.

Table A8.3: Mutale Shaft washability by size

RD	+6.7mm Material				-6.7+3.35mm Material			
	Fractional		Cumulative		Fractional		Cumulative	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
1.30			0.0	0.0	3.61	4.20	3.6	4.2
1.35	6.7	10.3	6.7	10.3	11.65	9.20	15.3	8.0
1.40	12.5	14.4	19.2	13.0	19.61	14.40	34.9	11.6
1.45	14.2	19.9	33.4	15.9	15.48	19.50	50.4	14.0
1.50	10.8	23.5	44.2	17.8	10.10	24.00	60.5	15.7
1.55	7.4	30.4	51.6	19.6	6.39	28.90	66.8	17.0
1.60	4.1	34.3	55.8	20.7	3.74	33.70	70.6	17.8
1.65	3.3	36.8	59.0	21.6	3.28	37.60	73.9	18.7
1.70	3.4	40.8	62.4	22.6	2.22	41.40	76.1	19.4
1.75	2.8	48.6	65.2	23.7	2.67	44.40	78.7	20.2
1.80	3.0	48.8	68.2	24.8	1.81	48.80	80.6	20.9
S1.80	31.8	74.2	-	-	19.45	71.70	-	-
Total	100.0	40.5	100.0	40.5	100.0	30.8	100.0	30.8
RD	-3.35+1.4mm Material				-1.4+0.6mm Material			
	Fractional		Cumulative		Fractional		Cumulative	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
1.30	14.9	3.7	14.9	3.7	24.2	2.7	24.2	2.7
1.35	15.6	8.7	30.5	6.3	22.0	7.0	46.2	4.8
1.40	17.7	13.6	48.2	9.0	13.4	13.7	59.6	6.8
1.45	11.8	19.2	60.0	11.0	7.8	18.7	67.4	8.1
1.50	7.3	23.7	67.3	12.4	5.0	23.4	72.4	9.2
1.55	4.3	28.7	71.6	13.3	4.4	29.5	76.8	10.4
1.60	3.6	33.1	75.1	14.3	1.9	30.3	78.7	10.8
1.65	2.8	37.0	77.9	15.1	2.5	32.7	81.2	11.5
1.70	1.6	41.4	79.5	15.6	1.5	33.5	82.7	11.9
1.75	1.9	44.4	81.4	16.3	1.8	35.8	84.5	12.4
1.80	2.0	49.5	83.4	17.1	1.4	41.2	85.9	12.9
S1.80	16.6	70.6	-	-	14.1	69.1	-	-
Total	100.0	25.9	100.0	25.9	100.0	20.8	100.0	20.8
RD	-0.6+0.212mm Material				-0.212+0.075mm Material			
	Fractional		Cumulative		Fractional		Cumulative	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
1.30	35.4	2.7	35.4	2.7	28.0	3.3	28.0	3.3
1.35	21.3	7.4	56.7	4.5	28.4	5.8	56.4	4.6
1.40	11.5	14.1	68.2	6.1	10.5	9.2	66.9	5.3
1.45	4.6	20.2	72.8	7.0	4.5	15.8	71.4	5.9
1.50	3.4	23.2	76.2	7.7	4.0	20.1	75.4	6.7
1.55	3.5	27.4	79.7	8.6	3.2	23.5	78.6	7.4
1.60	1.5	29.1	81.2	9.0	2.8	25.0	81.4	8.0
1.65	1.7	31.9	83.0	9.4	2.0	30.8	83.4	8.5
1.70	1.3	33.2	84.3	9.8	1.0	31.7	84.4	8.8
1.75	1.3	36.0	85.6	10.2	2.3	37.2	86.7	9.6
1.80	1.3	39.5	86.9	10.6	0.7	42.5	87.4	9.8
S1.80	13.1	72.6	-	-	12.6	67.3	-	-
Total	100.0	18.8	100.0	18.8	100.0	17.1	100.0	17.1

Table A8.4: Mutale Shaft washability by size

RD	+6.7mm Material				-6.7+3.35mm Material			
	Fractional		Cumulative		Fractional		Cumulative	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
1.30	0.0	0.0	0.0	0.0	2.9	3.7	2.9	3.7
1.35	3.6	9.4	3.6	9.4	10.8	8.5	13.7	7.5
1.40	10.6	13.6	14.2	12.5	21.6	13.5	35.3	11.2
1.45	16.5	20.9	30.7	17.0	17.5	18.3	52.7	13.5
1.50	12.2	23.2	42.9	18.8	10.8	23.2	63.5	15.2
1.55	6.9	30.1	49.8	20.3	6.7	29.0	70.2	16.5
1.60	4.2	33.2	54.0	21.3	3.8	33.1	74.0	17.3
1.65	3.1	34.5	57.2	22.1	3.0	37.7	77.0	18.1
1.70	2.9	44.2	60.0	23.1	2.2	41.1	79.2	18.8
1.75	2.2	44.4	62.2	23.9	2.7	44.5	81.9	19.6
1.80	2.5	48.0	64.8	24.8	2.2	48.3	84.1	20.4
S1.80	35.2	72.3	-	-	15.9	69.9	-	-
Total	100.0	41.6	100.0	41.6	100.0	28.2	100.0	28.2
RD	-3.35+1.4mm Material				-1.4+0.6mm Material			
	Fractional		Cumulative		Fractional		Cumulative	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
1.30	18.1	3.2	18.1	3.2	23.5	2.4	23.5	2.4
1.35	16.0	8.2	34.1	5.5	28.0	6.0	51.4	4.4
1.40	18.1	13.2	52.2	8.2	13.3	13.0	64.7	6.1
1.45	12.0	18.5	64.2	10.1	7.8	18.6	72.5	7.5
1.50	7.4	24.5	71.5	11.6	5.1	24.4	77.7	8.6
1.55	5.0	28.4	76.5	12.7	3.8	28.1	81.5	9.5
1.60	2.7	32.9	79.2	13.4	1.8	32.4	83.3	10.0
1.65	2.5	36.4	81.6	14.1	1.7	34.5	85.0	10.5
1.70	1.5	41.8	83.1	14.6	1.3	35.0	86.2	10.8
1.75	1.9	44.4	85.1	15.3	1.1	35.2	87.3	11.1
1.80	1.6	48.9	86.6	15.9	1.4	38.7	88.7	11.6
S1.80	13.4	68.3	-	-	11.3	68.3	-	-
Total	100.0	22.9	100.0	22.9	100.0	18.0	100.0	18.0
RD	-0.6+0.212mm Material				-0.212+0.075mm Material			
	Fractional		Cumulative		Fractional		Cumulative	
	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)	Yield (%)	Ash (%)
1.30	37.2	2.4	37.2	2.4	19.6	2.5	19.6	2.5
1.35	21.5	7.3	58.7	4.2	31.5	4.8	51.1	3.9
1.40	10.5	13.7	69.2	5.6	18.4	8.6	69.6	5.2
1.45	6.9	18.8	76.1	6.8	6.9	14.3	76.5	6.0
1.50	3.3	22.6	79.4	7.5	3.8	18.7	80.3	6.6
1.55	3.3	25.6	82.7	8.2	3.3	21.1	83.6	7.2
1.60	2.7	28.3	85.5	8.9	3.2	25.3	86.7	7.8
1.65	1.0	29.8	86.4	9.1	1.8	25.4	88.6	8.2
1.70	1.7	34.8	88.1	9.6	0.8	32.8	89.4	8.4
1.75	1.5	39.1	89.6	10.1	1.4	36.5	90.8	8.8
1.80	1.2	40.9	90.8	10.5	0.6	37.7	91.4	9.0
S1.80	9.2	69.7	-	-	8.6	66.9	-	-
Total	100.0	15.9	100.0	15.9	100.0	14.0	100.0	14.0

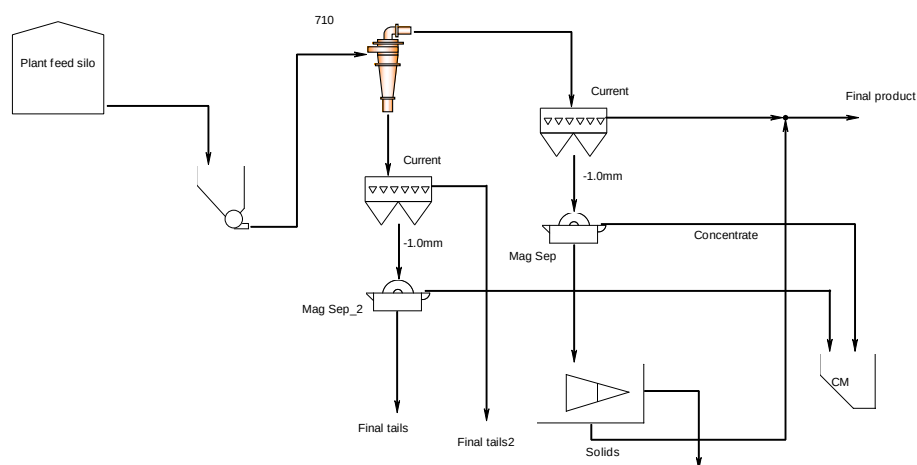
A8.2 Capital Estimates for the Flowsheet Scenarios

This section shows the estimated capital cost estimates for the reconfiguration of the plant according to the various options examined. Standard Exxaro factors for piping, instrumentation, electrical, civil, mechanical, installation, project management and contingencies have been included.

A8.2.1 Scenario1: All to 710mm diameter DMS cyclone

Table A8.5: Capital and installation costs scenario 1

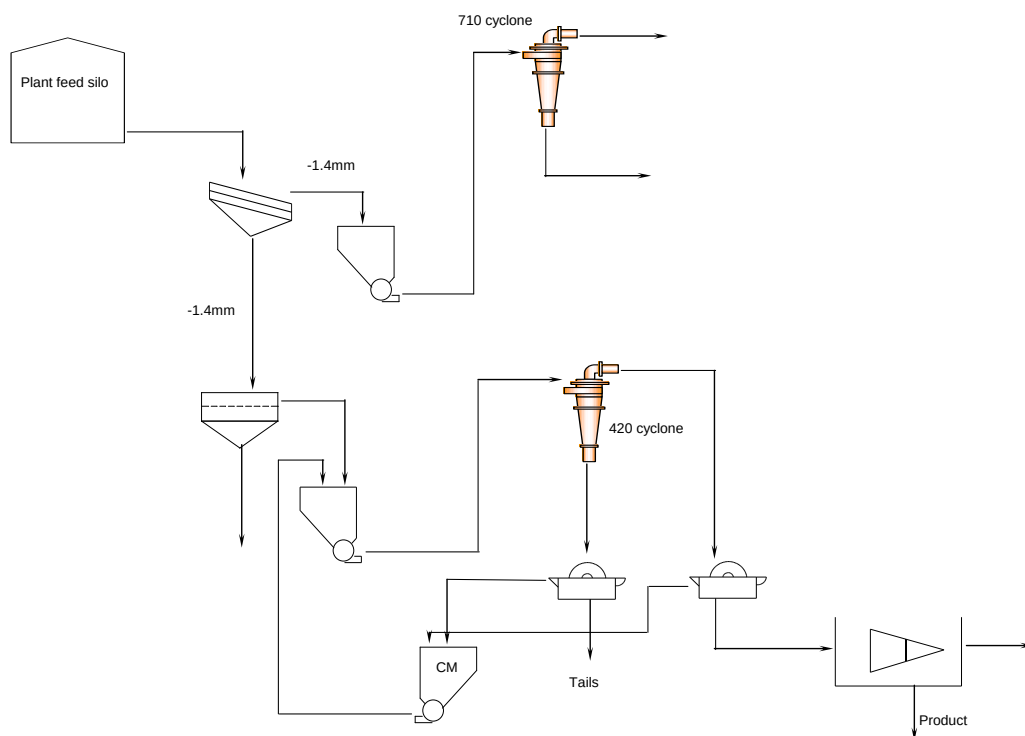
Name		Unit price	Quantity	Total
Pump + motor		120 000	-	-
Cyclones		80 000	-	-
Magnetic Seperator		250 000	4	1 000 000
Cyclone mixing boxes		100 000	1	100 000
Extra tank		130 000	2	260 000
Removal of old spirals		150 000	1	150 000
Removal of feed screen		100 000		
Dewatering screen		250 000		-
Sub Total		1 180 000		1 510 000
Piping	25%			377 500
Instrumentation	25%			377 500
Electrical	25%			377 500
Civil and mechanical	25%			377 500
Installation	25%			377 500
Project Management	20%			302 000
Contingency	30%			1 109 850
Total				4 809 350



A8.2.2 Option 2: +1.4mm to 710mm diameter DMS cyclone with the -1.4mm to 420mm diameter DMS cyclone

Table A8.6: Capital and installation costs scenario 2

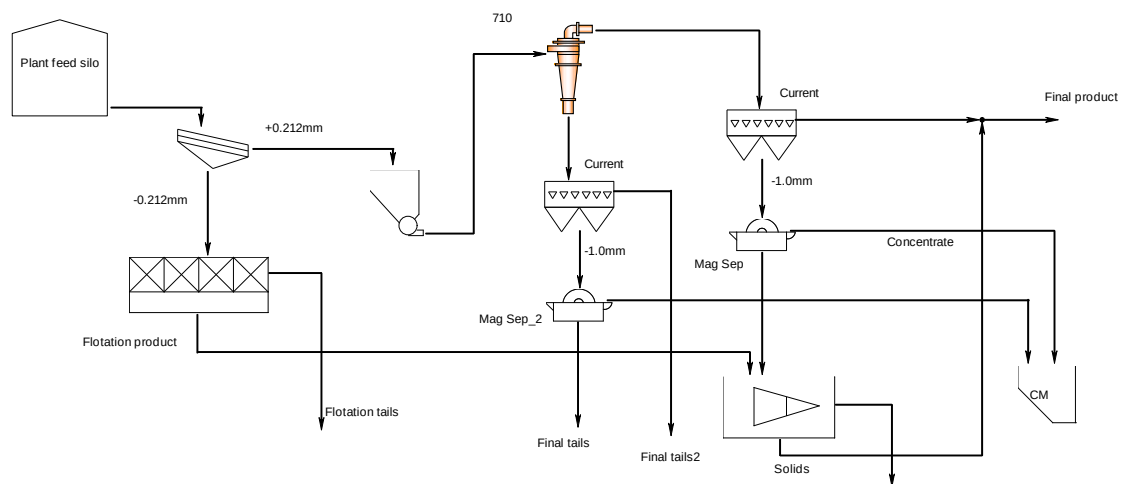
Name		Unit price	Quantity	Total
Pump + motor		120 000	4	480 000
Cyclones		80 000	2	160 000
Magnetic Separator		250 000	4	1 000 000
Cyclone mixing boxes		100 000	1	100 000
Extra tank		130 000	2	260 000
Removal of old spirals		150 000	1	150 000
Removal of feed screen		100 000	-	-
Dewatering screen		250 000		-
Sub Total		1 180 000		2 150 000
Piping	25%			537 500
Instrumentation	25%			537 500
Electrical	25%			537 500
Civil and mechanical	25%			537 500
Installation	25%			537 500
Project Management	20%			430 000
Contingency	30%			1 580 250
Total				6 847 750



A8.2.3 Scenario 3: +0.212mm material to 710mm diameter DMS cyclone with -0.212mm material to froth flotation

Table A8.7: Capital and installation costs scenario 3

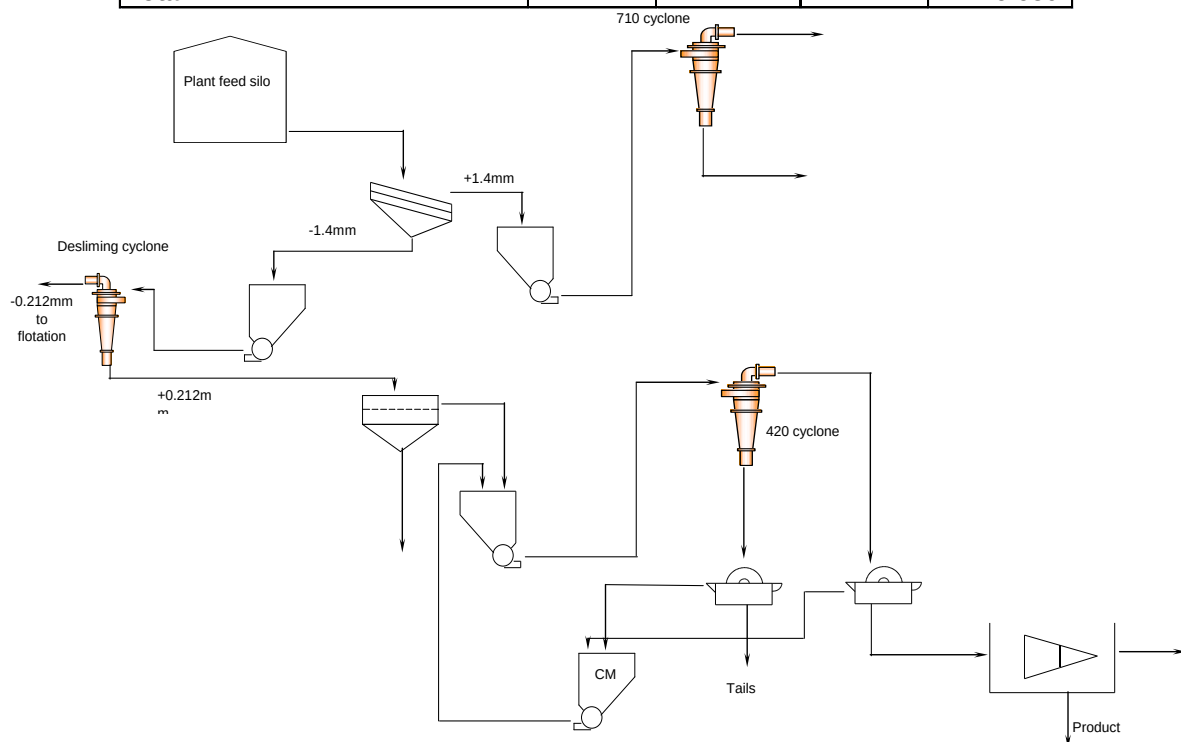
Name		Unit price	Quantity	Total
Pump + motor		120 000	-	-
Cyclones		80 000	-	-
Magnetic Seperator		250 000	4	1 000 000
Cyclone mixing boxes		100 000	-	-
Extra tank		130 000	2	260 000
Removal of old spirals		150 000	1	150 000
Removal of feed screen		100 000		-
Dewatering screen		250 000		-
Sub Total		1 180 000		1 410 000
Piping	25%			352 500
Instrumentation	25%			352 500
Electrical	25%			352 500
Civil and mechanical	25%			352 500
Installation	25%			352 500
Project Management	20%			282 000
Contingency	30%			1 036 350
Total				4 490 850



A8.2.4 Scenario 4: +1.4mm material to 710mm diameter DMS cyclones, -1.4+0.212mm material to 420mm diameter DMS cyclone, -0.212mm material to froth flotation

Table A8.8: Capital and installation costs scenario 4

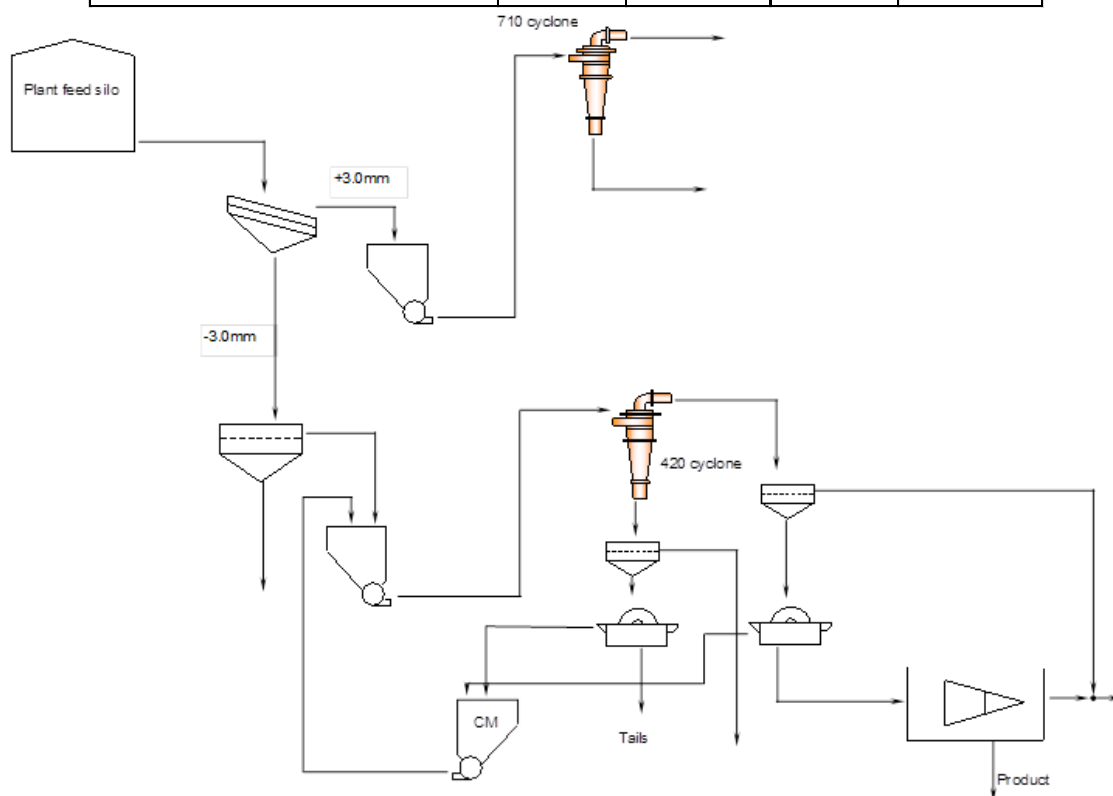
Name		Unit price	Quantity	Total
Pump + motor		120 000	5	600 000
Cyclones		80 000	2	160 000
Magnetic Separator		250 000	4	1 000 000
Cyclone mixing boxes		100 000	1	100 000
Extra tank		130 000	2	260 000
Removal of old spirals		150 000	1	150 000
Removal of feed screen		100 000	-	-
Dewatering screen		250 000		-
Sub Total		1 180 000		2 270 000
Piping	25%			567 500
Instrumentation	25%			567 500
Electrical	25%			567 500
Civil and mechanical	25%			567 500
Installation	25%			567 500
Project Management	20%			454 000
Contingency	30%			1 668 450
Total				7 229 950



A8.2.5 Scenario 5: +3mm material to 710mm diameter DMS cyclone with -3mm material to 420mm diameter DMS cyclone

Table A8.9: Capital and installation costs scenario 5

Name		Unit price	Quantity	Total
Pump + motor		120 000	4	480 000
Cyclones		80 000	2	160 000
Magnetic Separator		250 000	4	1 000 000
Cyclone mixing boxes		100 000	1	100 000
Extra tank		130 000	2	260 000
Removal of old spirals		150 000	1	150 000
Removal of feed screen		100 000	-	-
Dewatering screen		250 000	2	500 000
Sub Total		1 180 000		2 650 000
Piping	25%			662 500
Instrumentation	25%			662 500
Electrical	25%			662 500
Civil and mechanical	25%			662 500
Installation	25%			662 500
Project Management	20%			530 000
Contingency	30%			1 947 750
Total				8 440 250



A8.3 Comparison of LIMN and Coaltech Models

The results from LIMN were used in the financial evaluation to keep the comparison between the scenarios as constant as possible. The Coaltech simulation results were used to verify the LIMN results where possible. The LIMN results differed less than 1% yield from the Coaltech results for a constant product ash. The results are shown in Table A8.10 and Table A8.11 overleaf

Table A8.10: Vhukati Shaft simulation results

LIMN

	-212 Ash%	-212 Yield%	% of feed	Ash% -1.4	Yield -1.4	% of feed	Ash% +1.4	Yield +1.4	% of feed	Total Ash	Total yield	
Scenario 1: All to 710 cyclone										14.0	57.3	0
Scenario 2: -1.4mm to new 420 cyclone; +1.4mm to current 710 cyclone				11.6	75.6	28.3	15.5	50.3	71.7	14.0	57.5	100
Scenario 3: -0.212mm to flotation; +0.212mm to 710 cyclones	10.0	85.0	5.9				14.3	57.0	94.1	14.0	58.7	100
Scenario 4: -0.212 to flotation; -1.4+0.212 to 420 cyclone; +1.4mm to current 710	10.0	85.0	5.9	12.0	77.1	22.4	15.5	52.8	71.7	14.0	60.1	100
Scenario 4 comparison to Coaltech old model	8.8	82.0	5.9	14.0	83.7	22.4	14.0	41.1	71.7	13.5	53.1	100
Scenario 5: -3mm to 420mm cyclone, +3mm to 710mm cyclone				11.9	68.4	44.3	16.6	47.5	55.7	14.1	56.8	100

Limn - 1% yield lower than old model and 0.6% lower than new model

Coaltech - New model

	-212 Ash%	-212 Yield%	% of feed	Ash% -1.4	Yield -1.4	% of feed	Ash% +1.4	Yield +1.4	% of feed	Total Ash	Total yield	
Scenario 1: All to 710 cyclone										14.0	57.9	0
Scenario 3: -0.212mm to flotation; +0.212mm to 710 cyclones	8.8	82.0	5.9				14.0	56.2	94.1	13.7	57.8	100

Coaltech Old model

	-212 Ash%	-212 Yield%	% of feed	Ash% -1.4	Yield -1.4	% of feed	Ash% +1.4	Yield +1.4	% of feed	Total Ash	Total yield	
Scenario 4: -0.212 to flotation; -1.4+0.212 to 420 cyclone; +1.4mm to current 710	8.8	82.0	5.9	10.0	66.5	22.4	15.6	51.0	71.7	13.6	56.3	100
Scenario 4: -0.212 to flotation; -1.4+0.212 to 420 cyclone; +1.4mm to current 710	8.8	82.0	5.9	12.0	81.0	22.4	15.1	47.7	71.7	13.6	57.2	100
Scenario 4: -0.212 to flotation; -1.4+0.212 to 420 cyclone; +1.4mm to current 710	8.8	82.0	5.9	14.0	87.8	22.4	14.0	41.3	71.7	13.5	54.1	100

Table A8.11: Mutale Shaft simulation results

LIMN

	-212 Ash%	-212 Yield%	% of feed	Ash% -1.4	Yield -1.4	% of feed	Ash% +1.4	Yield +1.4	% of feed	Total Ash	Total yield	
Scenario 1: All to 710 cyclone										14.0	66.5	0
Scenario 2: -1.4mm to new 420 cyclone; +1.4mm to current 710 cyclone				11.0	81.9	34.3	16.3	59.0	65.7	14.0	66.9	100
Scenario 3: -0.212mm to flotation; +0.212mm to 710 cyclones	10.0	90.0	9.5				14.4	64.3	90.5	14.0	66.7	100
Scenario 4: -0.212 to flotation; -1.4+0.212 to 420 cyclone; +1.4mm to current 710 cyc	10.0	90.0	9.5	12.0	84.5	24.8	16.0	56.8	65.7	14.0	66.8	100
Scenario 4 comparison to Coaltech old model	8.4	86.8	9.5	13.9	90.2	24.8	14.1	45.5	65.7	13.2	60.5	100
Scenario 5: -3mm to 420mm cyclone, +3mm to 710mm cyclone				11.4	76.4	51.7	18.1	55.6	48.3	14.1	66.4	100

Limn - 0.8% yield higher than old model but 1% lower than new model

Coaltech - New model

	-212 Ash%	-212 Yield%	% of feed	Ash% -1.4	Yield -1.4	% of feed	Ash% +1.4	Yield +1.4	% of feed	Total Ash	Total yield	
Scenario 1: All to 710 cyclone										14.0	65.7	0
Scenario 3: -0.212mm to flotation; +0.212mm to 710 cyclones	8.4	86.8	9.5				14.0	63.2	90.8	13.5	65.6	100

Coaltech Old model

	-212 Ash%	-212 Yield%	% of feed	Ash% -1.4	Yield -1.4	% of feed	Ash% +1.4	Yield +1.4	% of feed	Total Ash	Total yield	
Scenario 4: -0.212 to flotation; -1.4+0.212 to 420 cyclone; +1.4mm to current 710 cyc	8.4	86.8	9.5	13.9	93.3	24.8	14.1	45.9	65.7	13.2	61.5	100
Scenario 4: -0.212 to flotation; -1.4+0.212 to 420 cyclone; +1.4mm to current 710 cyc	8.4	86.8	9.5	12.0	88.7	24.8	15.3	52.9	65.7	13.3	65.0	100
Scenario 4: -0.212 to flotation; -1.4+0.212 to 420 cyclone; +1.4mm to current 710 cyc	8.4	86.8	9.5	10.0	76.8	24.8	16.0	57.1	65.7	13.3	64.8	100